МЕТАЛЛУРГ

METALLURGIST

(METALLURG)

IN ENGLISH TRANSLATION

1957

NO. 2

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of the USSR

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SIGNIFICANCE OF ABBREVIATIONS MOST FREQUENTLY ENCOUNTERED IN SOVIET PERIODICALS

FIAN	Phys. Inst. Acad. Sci. USSR.
GDI	Water Power Inst.
GITI	State Sci.-Tech. Press
GITTL	State Tech. and Theor. Lit. Press
GONTI	State United Sci.-Tech. Press
Gosenergoizdat	State Power Press
Goskhimizdat	State Chem. Press
GOST	All-Union State Standard
GTTI	State Tech. and Theor. Lit. Press
IL	Foreign Lit. Press
ISN (Izd. Sov. Nauk)	Soviet Science Press
Izd. AN SSSR	Acad. Sci. USSR Press
Izd. MGU	Moscow State Univ. Press
LEIIZhT	Leningrad Power Inst. of Railroad Engineering
LET	Leningrad Elec. Engr. School
LETI	Leningrad Electrotechnical Inst.
LETIIZhT	Leningrad Electrical Engineering Research Inst. of Railroad Engr.
Mashgiz	State Sci.-Tech. Press for Machine Construction Lit.
MEP	Ministry of Electrical Industry
MES	Ministry of Electrical Power Plants
MESEP	Ministry of Electrical Power Plants and the Electrical Industry
MGU	Moscow State Univ.
MKhTI	Moscow Inst. Chem. Tech.
MOPI	Moscow Regional Pedagogical Inst.
MSP	Ministry of Industrial Construction
NII ZVUKSZAPIOI	Scientific Research Inst. of Sound Recording
NIKFI	Sci. Inst. of Modern Motion Picture Photography
ONTI	United Sci.-Tech. Press
OTI	Division of Technical Information
OTN	Div. Tech. Sci.
Stroiizdat	Construction Press
TOE	Association of Power Engineers
TsKTI	Central Research Inst. for Boilers and Turbines
TsNIEL	Central Scientific Research Elec. Engr. Lab.
TsNIEL-MES	Central Scientific Research Elec. Engr. Lab.—Ministry of Electric Power Plants
TsVTI	Central Office of Economic Information
UF	Ural Branch
VIESKh	All-Union Inst. of Rural Elec. Power Stations
VNIIM	All-Union Scientific Research Inst. of Meteorology
VNIIZhDT	All-Union Scientific Research Inst. of Railroad Engineering
VTI	All-Union Thermotech. Inst.
VZEI	All-Union Power Correspondence Inst.

Note: Abbreviations not on this list and not explained in the translation have been transliterated, no further information about their significance being available to us. — Publisher.

THE MAGNITOGORSK STEEL WORKS – 25TH ANNIVERSARY

Twenty-five years ago, in January, 1932 the first blast furnace was blown in at the Magnitogorsk steel works. At the foot of the Magnitnaya mountain, on the barren steppe, at an unprecedented speed, numerous factory buildings and installations sprang up, and a new metallurgical enterprise was initiated incorporating the very latest advances of modern technique.

Construction of the steel works, commenced in 1929, formed an integral part of gigantic works for setting up a new giant iron and coal base in the East – in the Ural-Kuzbass.

During the period of the Five-Year-Plans the Magnitogorsk steel works grew into a vast iron and steel concern, equipped with heavy plant machinery. The raw material for the steel works was provided by the reserves of rich iron ore in the Magnitnaya mountain together with the deposits of limestone, dolomite, fireclay and other nonmetallic minerals in the immediate neighborhood of these ores. A large reservoir was set up on the River Ural for supplying the steel works with water. Anthracite for coke production comes from the Kuznetsk and Karaganda coal fields; Karaganda coal is used for providing power.

Alongside the steel works grew the steel town Magnitogorsk. The steel workers live in comfortable houses. Schools, hospitals, public nurseries, a palace of culture and theaters have been built.

The Magnitogorsk steel workers are firmly promoting the development and introduction of modern technique and advanced technology.

In the first year of the sixth Five-Year-Plan the works team achieved a new labor victory. The production plan was fulfilled at every stage in the steel works program. The national target for ore-mining was fulfilled to 103.4%, for pig-iron to 101.1%, for steel to 100.6%, and for rolled steel to 100.1%. In 1956, as compared with 1955, 4.1% more ore was produced, 4.2% more pig-iron, 7.4% more steel and 6.4% more rolled steel.

Marked successes were achieved by the blast-furnace team. In recent years as a result of the coordinated introduction of a number of measures (delivery of raw materials, sharp reduction in the manganese ore content in the charge, changeover to the 7-8 iron tapping system, and so on) the blast-furnace workers achieved high productivity of the furnace.

Having fulfilled the 1956 production program ahead of time, the blast-furnace workers turned out tens of thousands of tons of pig-iron, in excess of the plan, effected a saving of 14 kg of coke for each ton of pig-iron and made a saving of more than 10 million rubles over and above the plan by careful expenditure of raw material, fuel and materials. The coefficient of the useful furnace volume in 1956 comprised 0.630 against 0.648 in 1955.

The best figures were achieved by the works team of No. 7 blast furnace under the foremen L. Ryabtsev, I. Kolduzov and K. Khabarov. Pig-iron production was in excess of the plan by 18.530 t, coke consumption per ton of pig-iron was reduced by 20 kg, the lowest mean annual coefficient of utilization of useful furnace volume in the country was achieved (0.607) and a saving of 2 million 450 thousand rubles was effected – these are the main working results of this team in the past year.

The Jubilee Year is marked by fresh labor successes by the steel smelting workers, while year by year the operation of the open-hearth furnaces is improved. Increase in the weight of the charge of the 180-ton furnaces to 200 tons and of the 380-ton furnaces to 400 tons, the complete changeover of all the furnaces to magnesite chrome arches, speeding up the charge by raising the volume of the mold from 1.24 to 1.75 m^3 , improved technology for conducting the smelt, the application of the most efficient methods of teeming the metal, improved maintenance of the equipment, extension of the campaign and reduction in the furnace stoppages for cold and hot overhaul: all this made it possible for the open-hearth furnace men to raise production and improve the quality of the steel.

The yield of steel per square meter of hearth in 1956 was 8.87 as against 8.70 tons in 1955. Stoppages were reduced to 7.1%. The over-all rejects from the open-hearth plants, including the rolling mills, were 0.49%

Examples of high labor productivity were provided by the smelters on the No. 12 open-hearth furnace (S. Bardin, G. Ozerov and G. Tatarintsev) who fulfilled the plan for the first year of the sixth Five-Year Plan ahead of time and turned out 7,540 tons of metal over and above the annual assignment.

The rolling-mill teams showed a marked increase in the volume of production, extended the range of rolled sections and improved the qualtity of production. This was helped by the increased size of the soaking pits on individual mills, increased rolling rates on No. 1 and No. 3 300-mills by changing the electric motors of the finishing stands for more powerful motors, the introduction of advanced technology, the mechanization and automation of the production processes, modernization of equipment and the introduction of technological improvements.

High figures were achieved by the medium sheet-mill team, rolling 6,550 tons of metal over and above the annual assignment.

Substantial and important tasks in the further increase in the production of pig-iron, steel and rolled steel lie ahead of the Magnitogorsk steel workers in the second year of the sixth Five-Year Plan.

It stands to the honor of the Magnitogorsk steel workers to go all out for the successful fulfillment of the tasks of the second year of the sixth Five-Year Plan and to register fresh labor victories.

BLAST-FURNACE PRODUCTION

IRON TAPPING FROM THE BLAST-FURNACE WELL IN THE LIQUID STATE

B. L. Tavrog
Blast-furnace Manager

E. D. Dmitrash
Chief Engineer, Engineering Department, Zaporozhstal Plant

The No. 3 blast furnace with a useful volume of 1300 m^3 was built in 1938 on the Gipromez prototype project and blown in after reconstruction on June 27, 1947. In June, 1951 the furnace was given an average overhaul and reconstruction and was turned over to operation with high gas pressure in the throat in April, 1952. The furnace operated without complete overhaul for 8 years, 6 months, 23 days, yielding 4,075,894 tons of pig-iron, including 12,500 tons of foundry iron.

On reconstruction of the furnace in 1947 the foundation was in a satisfactory condition. Only the disintegrated section of the reinforced concrete under the well was removed and replaced. The temperature of the foundation up to July, 1955 was held at a level of 260-320° and began then to rise, reaching 700° in December. On November 10, 1955 a crack-up to 10 mm was observed at the jacket of the hearth between tuyeres Nos. 1 and 2 at a height of approximately 1.5 m.

The crack was welded up. In December, 1955 a crack again appeared at the point of the welded joint and this was again welded up.

Weak combustion of the gas was observed below the well jacket.

In view of the sharp rise in the foundation temperature the furnace was put on to smelting lower-temperature pig-irons with a silicon content up to 0.3-0.4%, manganese up to 2% and less, and with an increased sulfur content. The number of tappings was increased to seven. The gas pressure under the throat was reduced. On December 19, 1955 the furnace was put on to low pressure.

Despite these measures the temperature of the foundation rose from 700 to 760° during the period from December 7-19.

Further steps were taken: the coke charge was raised to 8.8 tons and then to 10 tons, employing the charging system CCCICAAx. Close observation was maintained of the state of the hearth and its cooling.

Further temperature rise of the foundation was checked.

On January 18, 1956 the furnace was stopped for complete overhaul.

In order to achieve the most complete tapping of the iron from the well in the liquid state and for maximum utilization it was decided to carry out tapping through two bear notches: on the pig-iron side at 2200 mm below the axis of the iron notch and on the slag side at 4400 mm below the iron notch.

For the upper notch an opening 500×500 mm was cut into the brickwork in the jacket and cooling box of the hearth the day before the blow-out. Brick was rammed into the brickwork to a depth of 600 mm and the jacket was carefully made up.

For the lower notch a trench was cut in the foundation with pneumatic hammers through two weeks. A 400 mm layer of concrete was preserved up to the refractory brickwork.

A refractory brick trough was laid on the work space in front of the upper notch, with a fork for pouring the iron into two ladles.

A refractory trough was set up inside the concrete in the trench of the lower notch with a lined iron nozzle fitted at the end.

The upper notch was intended for tapping the iron into 80-ton pig ladles (10 ladles were prepared); for tapping the metal through the lower notch 13 lined metal wagons were prepared (Fig. 1) with a capacity of 40 tons.

In order to avoid the iron being lost on the way during transport of the ladles, lined troughs were set up between the ladles and also between the 40-ton wagons.

Before blowing-out, open-hearth pig-iron was melted in the furnace with a composition 0.7-0.8% Si, approximately 2.5% Mn and 0.05-0.06% S.

The ore charge was reduced from 2.25-2.30 to 2.0 before blowing-out; dead coke charges were fed into the furnace with an over-all weight of 164 tons and at 11:15 P. M. on January 17th, charging was stopped.

Starting at 2 a.m. on January 18th, the normal blast of 2350 m^3/min was gradually reduced to 1500 m^3/min and the pressure was reduced to 0.3 atm. At 6:45 A. M. the last tapping was made, amounting to 112 tons. The iron contained 1.15% Si, 2.48% Mn and 0.068% S.

The blast pressure was maintained at 0.25 atm. from 7 A. M. to 10 A. M., at 0.1 atm. from 10 A. M. to 1 P. M. and subsequently at 0.05 atm at a rate of 500 m^3/min and a temperature of 800°.

At the end of the third shift on January 17th, water was delivered to the gas outlet and under the bell. The water pressure (5 atm) was gradually increased to 9 atm. by 1 A. M. on January 18th.

The throat temperature in the first shift of January 18th was 400-500° and in the second shift 400-200°.

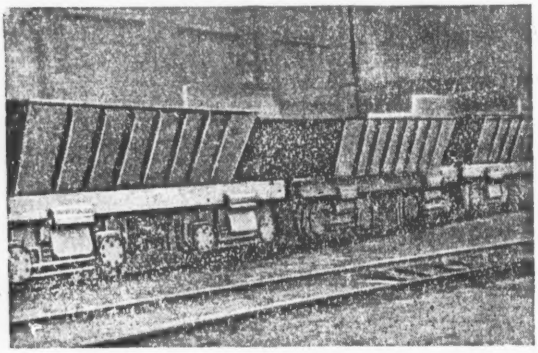


Fig. 1. Wagons for taking off pig-iron from the bottom bear notch.

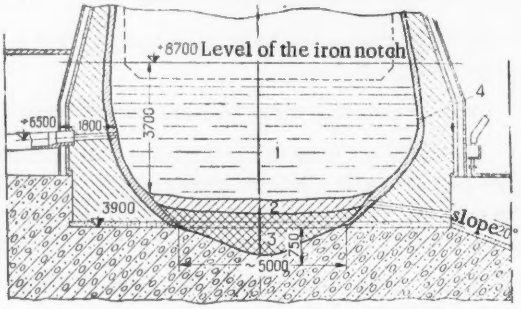


Fig. 2. Well erosion:
1) pig iron; 2) conglomerate consisting of metal, slag and refractories; 3) solid bear; 4) lining.

At 2 A. M. on January 18th work was commenced on opening up the lower bear notch. The opening was first drilled to a depth of 400 mm with a pneumatic drill; after burning out the opening with oxygen to a depth of 800 mm the iron came through. Four oxygen tubes were expended in the burn-through.

At 9:45 A. M. the first iron ladle began to fill and by 2:30 P. M. eight ladles were filled.

The pig-iron temperature at 1:20 A. M was 1260° and at 1 P. M. 1300°. The pig-iron had a low silicon content.

After bringing up the second ladle to the iron notch the first ladle was hauled off by loco. The connecting trough joining the first and second ladles was taken up by track crane. The remaining pig ladles were transported in the same sequence.

Burning out of the lower bear notch was rendered difficult by the fact that the notch was situated at the bottom of the trench. After expending six oxygen tubes a thin stream of iron was observed. The first three wagons were filled after 2 1/2 hours. After burning out an opening slightly below the first,the iron was more free-running and the stream was broader, another 7 wagons being filled by 11 P.M. By 11.30 P.M. 10 wagons were filled. Tapping was then completed.

In all,1047 tons of pig-iron were tapped, comprising 656 tons from the top notch and 391 tons from the bottom notch. The chemical composition of the pig-iron was as follows:

Sample number	Si	Mn	S	Sample number	Si	Mn	S
1	0.3	0.75	0.056	8	0.48	1.4	0.082
2	0.3	0.75	0.058	9	0.84	1.98	0.064
3	0.3	0.65	0.088	10	0.88	2.05	0.067
4	0.26	0.65	0.102	11	0.81	1.89	0.066
5	0.27	0.8	0.120	12	0.24	0.6	0.068
6	0.4	1.17	0.102	13	0.26	0.59	0.063
7	0.49	1.2	0.08	14	0.23	0.75	0.080
				15	0.32	0.72	0.075

After tapping the iron and skimming the coke, slag and conglomerate residues, the dimensions of the hole in the hearth were determined (Fig. 2). The depth of the hole was 3700 mm below the axis of the iron notch. On the bottom of the pocket there was a coagealed lining and conglomerate consisting of a mixture of pig-iron, slag, graphite and refractories. Veins of iron penetrated into the concrete of the foundation to a depth of 750mm. A solid bear weighing approximately 170 tons was removed by blasting.

Removing the pig-iron in the liquid state from the well, afforded a considerable reduction in the time taken for the overhaul.

IMPROVED FURNACE-CHARGING PROGRAM

N. I. Savichev
Magnitogorsk Steel Works Blast-Furnace Plant Foreman

The Magnitogorsk blast foremen assign prime importance to improved utilization of the thermal and chemical energy of the gas flow in raising productivity, lowering coke consumption and improving other technical economic performance indices of the furnaces.

In the blast-furnace changeover to operation with high gas pressure in the throat the foremen encountered a number of difficulties such as exposure at the periphery and intensification of the gas flow around the periphery, which is undesirable from the standpoint of complete utilization of the chemical energy of the gas. In order to load up the periphery, more efficacious measures were necessary.

The Zaporozhstal sintering hands are continually improving production, extending automatic control to the entire plant. Sintering hand Ivan Voinovsky is shown at the controls of a completely automatic sintering machine.

Photograph by N. V. Sidoruka.

According to the literature, in order to increase the charge at the periphery it was necessary to reduce the ore charge. Operating experience showed that reducing the ore charge below certain limits did not yield the required effect from the standpoint of charging up the periphery, since the ore settled out around the periphery in a very narrow ring while the periphery gasflow continued unaltered.

At the Magnitogorsk blast furnaces the quantity of ore-charge fed at one time from the large bell was kept constant within the range 14-18 tons.

Operation with high gas pressure in the throat allowed a higher furnace driving rate. The higher furnace driving rate, however, was obtained mainly by an increased ore charge.

The foremen on No. 8 blast furnace tried out a number of charging systems involving varying the size of the charge. In particular, a separate system was tried out with an 18.5 ton charge. In this system the ore charge was placed in a ring with open periphery and center. The charging system OCxOCCx was tested. With this system the efficiency of the gas was improved and the ore charge was increased, but on account of the increased charge the distribution of the gas flow varied sharply and channelling was intensified.

In the opinion of the Magnitogorsk foremen, in order to avoid channelling and to improve loading of the periphery it is much better to employ the OCOCC system. This system allows the periphery to be loaded up with a broad ring while maintaining sufficient activity in the center, and obviating channelling while improving descent of the charge.

It was observed that the more fines there were in the ore charge the greater must be the quantity of ore charge. Under these conditions it was necessary to work on the pressure-drop value, which was held within the range 1.20-1.25 atm.

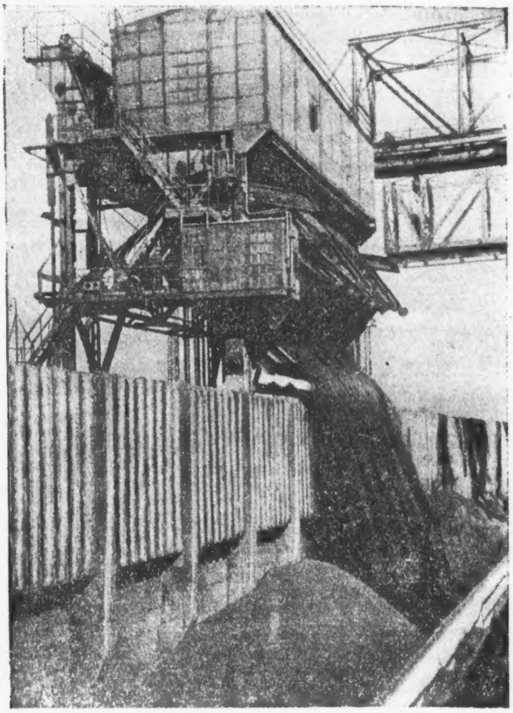
Employing these measures,it was possible to raise the ore charge to 2.70.

RECONSTRUCTION OF NO. 2 BLAST FURNACE

Engineer N. V. Krepyshev

Reconstruction of blast furnace No. 2 was carried out from July 17th to September 8th, 1956. By reason of the reduced thickness of the lining, the volume was increased from 1163 to 1310 m^3.

The increased volume of No. 2 blast furnace rendered it necessary to reinforce the foundation. In addition, a number of measures were carried out directed toward improving the furnace performance and lengthening the duration of the campaign.



The ore pile at the Zaporozhstal plant is completely mechanized. Three ore ladle cranes are employed in delivering the charging material. A high productivity discharge equipment is employed involving a wagon tippler which greatly expedites discharge of the materials. A 60 ton wagon is now discharged in two minutes.

Photograph by N. V. Sidoruka

In order to avoid the formation of a deep pocket, in the well and the occurrence of thermal stresses in the foundation, air cooling was installed under the well mounted on the bottom of the foundation housing on a layer of heat-resisting concrete. The underneath cooling took the form of a bundle of solid-drawn steel tubes of 100-150 mm diameter, the ends of which were drawn out around the foundation through an opening in the wall of the housing. The air for cooling the well was delivered by two Sirokko No. 10 blowers with a capacity of 16-18000 m^3/h. After two months operation on the furnace the air temperature drop was equal to 3-4°.

The lining of the lower part of the furnace was made up in the following way. In the lower section of the well two rows of carbon blocks were laid carrying six rows of semi-arch fireclay hearth brick type D2. Above this the peripheral section of the well and the walls at the metal level (up to the level of the slag notches) were lined with carbon blocks. The upper central section of the well was made up of three rows of large fireclay brick 400 mm high. The large brick in this section of the well was employed for convenience of joining up with the carbon blocks placed along the periphery close up to the end coolers.

The bosh lining was also made up of carbon blocks and the lower section of the blast-furnace body of carbonized fireclay brick.

A 3 ton jib crane was mounted on the lentil girder for facilitating the furnace work.

In addition an electric screw tap-hole gun with a piston pressure of 160-200 tons was installed providing closure of the iron notch during full driving of the blast furnace without reducing the blast and without lowering the gas pressure in the throat.

A distributor with combined packing was fitted in the furnace throat. Enclosed dip-rods were installed, obviating gas leakage. In order to provide quick disconnection of the furnace from the gas main during short stoppages, the gas-water slide valve was replaced by a cut-off valve at the primary dust catcher.

Operation of the air heaters and all the operations for charging the furnace including conveyance, weighing and transport of the raw materials to the skip bay and weigh cars were rendered automatic. Reconstruction of the furnace was carried out in fifty-four days.

(Information Bulletin No. 10 (16), Kuznets
Metallurgical Combine)

STEEL-SMELTING PRODUCTION

DE-SULFURIZATION OF TRANSFORMER STEEL OUTSIDE OF THE FURNACE

D. D. Burdakov

Manager, Technical Section, Glavuralmet, U.S.S.R. Ministry of Iron and Steel

At the Upper Iset steel works transformer steel is smelted in electric furnaces and basic open-hearth furnaces. Open-hearth furnaces with a charge of 90-100 tons are heated with oil. The quality of the transformer steel is determined by the specific power losses, which depend on the purity of the metal with reference to the additions, carbon, sulfur, phosphorus and nonmetallic inclusions. Sulfur has a particularly injurious effect on the electro-magnetic characteristics of transformer steel.

TABLE 1

Relationship of Power Losses to Sulfur Content in Transformer Steel

Number of smelts	S content %	Mean specific power losses p_{10}
39	up to 0.009	1.42
62	0.009-0.010	1.43
62	0.010-0.011	1.44
77	0.011-0.012	1.45
58	0.012-0.013	1.48
27	0.013-0.014	1.49
8	0.014-0.015	1.50

Table 1, drawn up on the basis of 333 smelts, shows that increasing sulfur content is accompanied by increasing power losses in the transformer metal. For high quality transformer steel smelted in the electric furnace the sulfur content must not exceed 0.005%.

In smelting in open-hearth furnaces an attempt is made to obtain transformer steel approximating both in chemical composition and in the quantity of nonmetallic inclusions to the metal smelted in electric furnaces. The chemical composition of transformer steel in % is set out below:

	C	Mn	Si
Electric furnace	up to 0.06	up to 0.15	4.00-4.20
open-hearth furnace	0.07	up to 0.30	3.96-4.25

	P	S	Al
	not more than		
electric furnace	0.015	0.005	0.05-0.12
open-hearth furnace	0.030	0.012	0.05-0.12

The technological process of smelting transformer steel in open-hearth furnaces is standardized. The main features are as follows:

1) The limestone in the charge comprises 10-12% of the weight of the metal charge, depending on the number of silicon steel taps (dynamo and transformer);

2) The first slag (1.5-2.0 m^3) is taken off 1-1.5 hours before complete smelting of the charge;

3) After full smelting of the charge in the bath, ore is introduced and slag is run off a second time (1-1.5 m^3);

4) The duration of the finishing boiling is 1.5 hours;

5) On reaching the required chemical composition the metal is held in the furnace without feeding fuel for 5-10 minutes and is then drawn off. Particular attention is paid to the heating temperature of the metal. Before drawing off the smelt, the temperature of the metal must be within the range 1610-1650° and in the ladle before teeming 1590-1630° (as according to immersion thermocouple).

It is known that for successful de-sulfurization of the metal in the open-hearth furnace it is necessary to have have charging materials and fuel free from sulfur, to obtain active slag of high basicity at the commencement of smelting of the charge and to renew the slag during the boiling process of the bath. Hot driving of the furnace and intense boiling of the bath provide high heating of the metal as a result of which the manganese content is raised as a result of reduction from the slag.

Transformer steel with a sulfur content below 0.012% can, however, be obtained in the open-hearth furnace only with prolonged reduction.

At the Upper Iset plant de-sulfurization of the transformer steel is carried out outside of the furnace by treating of the liquid metal with a powder mixture in the ladle during tapping. The mixture consists of 80% freshly calcined lime and 20% ground fluorspar. The consumption of the mixture per smelt is 1.1-1.2% of the weight of the metal part of the charge.

Approximately 30% of the mixture is put into the ladle before bringing under the smelt and 70% is introduced from the bunker into the trough along the path of the metal from the furnace. The whole of the mixture must be introduced into the metal before the slag starts to leave the furnace.

Before introducing the method of de-sulfurization outside the furnace at open-hearth furnaces, the mean sulfur content in transformer steel for all the smelts poured during 1951 was 0.0244% and in 1954, when treating all the smelts with the desulfurizing mixture, 0.0117%. On introducing the method of de-sulfurization outside of the furnace, the sulfur content in the steel could be reduced to half.

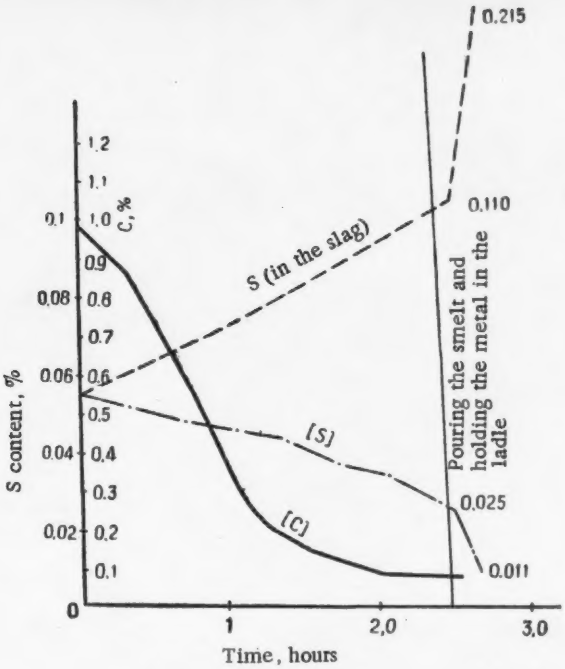
On treating the metal with the powder mixture for 15-20 minutes (comprising 5-8 minutes for pouring the smelt and 10-12 minutes with the metal in the ladle) there is a sharp reduction in the sulfur content in the metal and a rise in the sulfur content of the slag (see diagram).

In order to determine the influence of various ratios of lime and fluorspar and the quantity of mixture on the degree of de-sulfurization of the metal, a number of experimental smelts were carried out. The results of some of these tests are given in Table 2.

TABLE 2

Relationship of the Degree of De-sulfurization of the Metal and the Quantity and Composition of the Powder Mixture

Quantity of mixture						S content, %			De-sulfurization in the ladle	% sulfur reduction in the ladle
Lime		Fluorspar		Total, kg	% of metal part of the charge	During smelting	Before teeming	In the ingots		
kg	%	kg	%							
1200	80.0	300	20.0	1500	1.58	0.039	0.025	0.010	0.015	60.0
1100	73.0	400	26.7	1500	1.57	0.040	0.020	0.008	0.012	60.0
1000	66.7	500	33.3	1500	1.55	0.046	0.028	0.011	0.017	60.05
1400	82.4	300	17.6	1700	1.8	0.046	0.017	0.010	0.007	41.0
1500	83.3	300	16.7	1800	1.9	0.036	0.026	0.011	0.015	58.0
2000	91.0	200	9.0	2200	2.34	0.039	0.020	0.009	0.011	55.0



Burn-out of carbon and sulfur in the process of finishing and drawing-off the smelt.

On the basis of a large number of tests and industrial application of this method it was established that treatment of transformer steel in the ladle with a powder mixture consisting of freshly calcined lime and ground fluorspar considerably reduces the sulfur content of the metal. The best results are obtained employing a mixture consisting of 20-25% ground fluorspar and 80-75% of freshly calcined lime. The consumption of the mixture must not exceed 1.2-1.5% of the metal part of the charge.

U.S. S. R. STEEL SMELTERS' CONGRESS

THE PROBLEM OF MECHANIZATION AND AUTOMATION OF THE OXYGEN SUPPLY TO THE ELECTRIC STEEL-SMELTING FURNACE

G. S. Selkin (Central Iron and Steel Scientific Research Institute)
N. P. Zadalya (Zaporozhstal Open-Hearth Furnace Plant)

The question of the most effective method of introducing oxygen into the bath of open-hearth and electric furnaces was never so acute during the period of application of the technology. Large-scale steel production employing oxygen is considerably retarded through the lack of an ideal device for introducing oxygen into the metal. The use of iron tubes with or without insulation of the outer surface for blowing oxygen into the bath has proved itself for the conditions of small-scale steel smelting. This method cannot be recommended however, for regular operation on large furnaces. In addition, the demand on iron tubes is so great that the tube industry is not in a position to meet it. At only one Zaporozhstal plant, approximately 360 km of iron tubes would be required each year for blowing the baths of open-hearth furnaces with oxygen. The Dneprospetsstal electric furnaces require for the same purpose 200-250 km of tubes per year.

Hence, the search for the best method of introducing oxygen into the bath represents an important problem demanding early solution from the thermal engineering designers and steel smelting research specialists.

The Central Iron and Steel Scientific Research Institute has been carrying out experiments since 1953 at the Zaporozhstal plant for the purpose of investigating existing methods and developing new methods for introducing oxygen into the bath of open-hearth furnaces.

On the initiative of the workers at the plant and the CISSRI, a device was constructed for introducing oxygen into the bath through the arch of the open-hearth furnace. In the early experimental smelts this method of blowing showed considerable advantages, such as simplicity of design, ease of observation of the process, possibility of adding ore into the furnace during blowing and the possibility of full mechanization and automation of the oxygen supply to the bath.

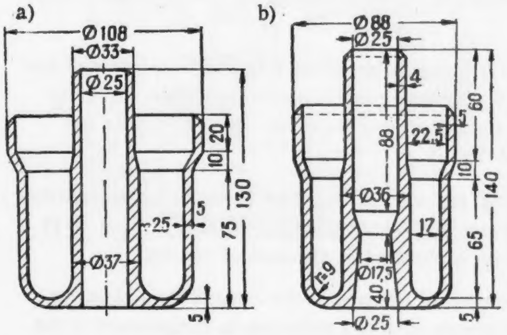


Fig. 1. Copper head designs for the water-cooled tuyeres:
A) CISSRI-1, with cylindrical aperture for the oxygen; B) CISSRI-2, with nozzle.

At the present time oxygen is delivered to the bath of open-hearth furnaces at the Zaporozhstal plant by means of a fully automatic arch construction, the main part of which consists of water-cooled tuyeres (a design developed on the instruction of the Laboratory of Metallurgical Thermal Engineering of the CISSRI by the Zaporozh sector of the TPKB and the Iron and Steel Power Trust).

Experimental operation with tuyeres of various design (Fig. 1) showed that with the head up to 200 mm above the slag level intense atomizing of the slag and metal occurs, large quantities of agitated smoke are smoke are formed and the coefficient of utilization of the oxygen is reduced. After completing blowing at a height of 0.3-0.7 m there is a buildup of slag and metal on the tuyere which impedes withdrawal of the tuyere from the furnace.

Calculations indicate that, with an oxygen pressure of 11 atm. and with the head above the slag level, a jet of oxygen 25 mm diameter enters the bath to a depth of 400 mm.

The depth of immersion H (mm) of the oxygen jet into the metal with various pressures P and immersion of the tuyere to a depth a under the slag level can be determined from the formula:

$$H = 120\sqrt{P} - a.$$

On immersing the nozzle in the bath to a depth up to 250 mm, spattering is diminished, but on blowing metal with a high carbon content (over 0.8%) it is still considerable and has a marked influence on the wear of the furnace lining.

In order to obviate spattering of the slag and metal on introducing oxygen into the bath, two variants of multi-jet water-cooled tuyere designs were developed and tried out under industrial conditions (Fig. 2). Fig. 3 shows the nature of a five-jet stream on a hydraulic model; four jets on leaving the tuyere form a helical jet, while the fifth is directed vertically into the metal.

For normal operation of the five-jet tuyere, contact is necessary with the surface of the bath. Under these conditions the blowing process proceeds steadily, without pulsations. With the introduction of the spiral jet into the bath, spattering of the slag and metal are absent.

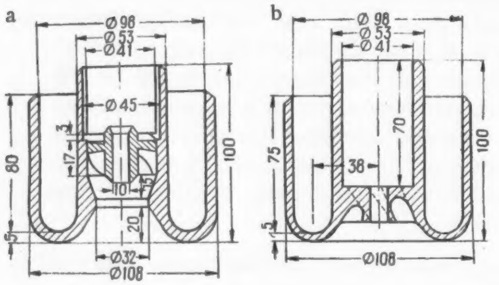


Fig. 2. 5-jet head designs for water-cooled tuyeres: A) CISSRI-3; B) CISSRI-4.

Comparison of the methods for blowing the metal with oxygen (with identical oxygen consumption and initial carbon content in the bath) indicates that on blowing the bath with the multi-jet tuyeres the duration of the final smelt is less than on blowing with tuyeres with cylindrical openings or with the usual iron pipes (Fig. 4). It can be assumed that the multi-jet tuyere raises the degree of utilization of the oxygen introduced into the metal and improves mixing of the slag and metal as compared with the cylindrical tuyere.

On blowing pure oxygen into the metal, very high temperatures are developed in the mouth of the tuyere, reaching 2200-2300°.

Results of investigations on the thermal regime of the tuyere submerged in the slag and metal showed that the system of cooling the five-jet tuyere, as developed, suits the conditions of its operation. Thus, the temperature at the bottom of the tuyere during the blowing period fluctuates over the range 150-200°, while the temperature difference of the water entering and leaving is 18-25°.

It was found at the Zaporozhstal plant that water entering into the blowing zone (even in large quantities) does not give rise to explosions or spitting. The presence of water in the bath gives rise to surges which, as it were, strive to eject the tuyere from the bath. The phenomenon is observed in all cases of burning.

At the present time tests are being carried out on an open-hearth furnace at the Zaporozhstal plant for blowing the bath with an oxygen-water mixture. Wetting the oxygen affords a reduction in temperature in the reaction zone as a result of which the blowing process proceeds without intense burn-up of the iron. As compared to blowing with pure oxygen the dust carry-over is reduced to a seventh or a tenth and the conditions of the thermal regime of the tuyere operation are improved.

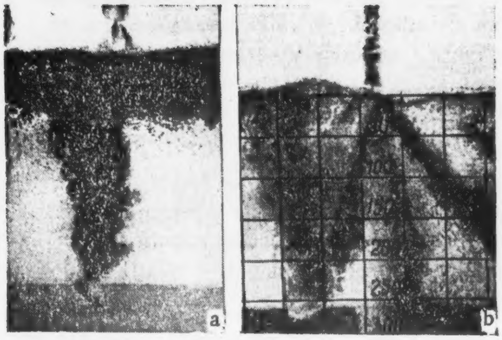


Fig. 3. Nature of the jet of a 5-jet head demonstrated on a hydraulic model:
a) CISSRI-1-head; b) CISSRI-4.

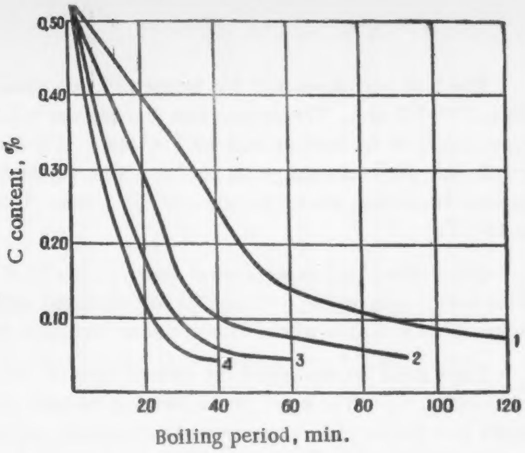


Fig. 4. Rate of carbon burn-up in the final melt period: 1) without using oxygen; 2) blowing the bath with oxygen employing two insulated pipes; 3) delivering oxgen in jet with subsequent blowing of the bath by means of two water-cooled tuyeres with cylindrical aperture; 4) ditto, by means of two water-cooled tuyeres in a 5-jet stream.

In the case of experimental IKh18N9T stainless smelts in the KMK* 30-ton electric furnaces a method was tried out for introducing oxygen into the bath by means of water-cooled tuyeres through the furnace arch as in the Zaporozhstal open-hearth furnace experiment. In the case of these smelts the tuyere was held in the clamp of one of the electrode holders (the electrode being removed). This permitted the tuyere to be lowered and raised by means of the mechanism for lowering and raising the electrode, from the control desk.

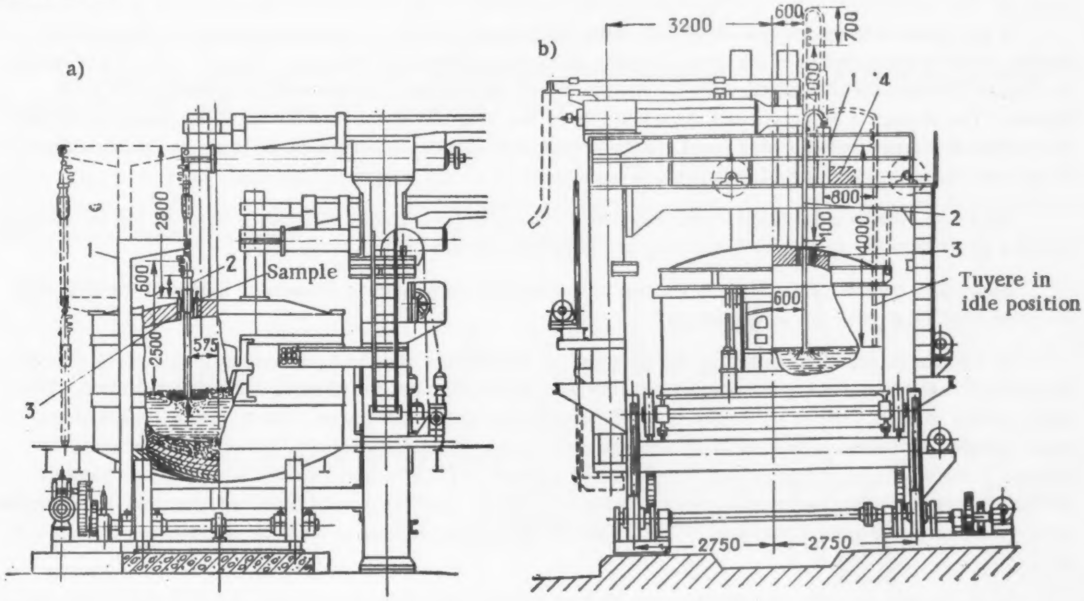


Fig. 5. Diagram of the arch construction for introducing oxygen into the bath of an electric furnace:
a) end-charging furnace; b) top-charging furnace.

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* KMK = Kuznets Metallurgical Combine — Publisher's note.

The bath was blown with the tuyere 200 mm above the slag level and with the tuyere immersed in the bath to 100-200 mm. The oxygen line pressure was held within the range 5.5-6.5 atm. corresponding to an oxygen supply to the bath of 14.8-18.3 m^3/min. The oxygen burning rate in the experimental smelts was: with blowing above the slag, 0.6% per hour; with tuyere immersed in the slag, 0.76% per hour. The water consumption for cooling the tuyere was 550-650 l/min. The temperature-drop of the water entering and leaving was 15-18°.

The results of the experimental smelts on the KMK electric furnaces show that when employing the water-cooled arch oxygen tuyere, it is possible to mechanize and apply automatic control to the blowing, to expedite the process of oxidation of the carbon and to eliminate disturbances in servicing the furnace itself.

The CISSRI has developed the general lines of an arch construction for introducing oxygen into arc furnaces (Fig. 5). The structure for blowing the bath with oxygen is set up on each furnace individually and consists of a hollow plain or compound (telescopic) movable bracket 1, from which the tuyere is suspended, the water-cooled tuyere 2, the water-cooled arch caisson 3, an electric drive for raising and lowering the water-cooled tuyere 4, the measurement and control gear and the oxygen and water lines.

The water-cooled arch caisson is made in various dimensions. For a tuyere with diameter 100 mm the caisson has outer dimensions of 350× 350 with an internal orifice diameter 250 mm. For a tuyere with diameter 75 mm the outer dimensions of the caisson are 300×300 mm with internal orifice diameter 200 mm.

Oxygen blowing of the bath is carried out by the steel smelter or his assistant by push-botton control according to the data of the blowing program.

Results of test smelts in a 30-ton arc furnace have demonstrated the complete reliability of automatic operation of the oxygen-blowing process through the furnace arch.

At present the technical requirements have been developed for designing an automatic arch construction.

Remote automatic control will be applied to lowering and raising the tuyere.

If the pressure in the oxygen main falls below the standard, if the maximum temperature drop of the cooling water is exceeded or if the oxygen supply is interrupted through blockage of tuyere with slag or metal, the tuyere is automatically raised. In addition the tuyere can be raised or lowered by operating the push-buttons. The oxygen is automatically connected in on lowering the tuyere and is cut off on raising the tuyere to a height 300 mm above the slag level. Introduction of oxygen is accompanied by lamp and buzzer signals. Lamp signals are provided for each position of the tuyere.

An oxygen meter is included in the automatic control system, and by this means blowing can be carried out to a given quantity of oxygen depending on the carbon content of the metal.

Automatic control of the blowing process of the metal in the electric furnace by means of the arch construction must be carried out in two stages.

In the first stage, after setting up the structure on the furnace and the measurement and control gear on the control desk and adjusting the operation of the gear and equipment, the blowing regime is finalized. The steel smelter manually opens up and shuts off the oxygen and raises and lowers the tuyere by means of the push-buttons. The blowing process is carried out according to the given operating parameters of the tuyere (oxygen pressure, position of the tuyere in relation to the level of the bath and so on), which are checked on indicating and recording instruments. After setting up the best operating condition of the arch construction with reference to the given conditions of operation, the automatic control of the arch construction is switched in.

In the second stage the blowing is controlled by means of the push-bottons according to the given quantity of oxygen with automatic opening up and shutting off of the oxygen and automatic raising of the tuyeres in case of failure.

U. S. S. R. STEEL SMELTERS' CONGRESS

THE TEMPERATURE MEASUREMENT OF LIQUID STEEL BY MEANS OF IMMERSION THERMOCOUPLES

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As a result of investigations carried out at the Kuznetsk-Metallurgical Combine (KMK) a platinum-platinum-rhodium immersion thermocouple has been developed together with a method for measuring the temperature of liquid steel in open-hearth and electric steel smelting furnaces.

The measuring equipment (Fig. 1 and 2) which is set up on each steel smelting furnace of the steel works comprises a valve potentiometer EP-107, mounted on the temperature panel of the furnace; two junction boxes, No 1 for measuring the temperature of the metal in the furnace, set up on the working space opposite the middle charging vent, and No. 2 for measuring the temperature of the metal in the ladle, set up on the barrier over the working space at the back end of the furnace. Compensating leads are run in tubes for connecting up the valve potentiometer to the junction boxes. The movable immersion platinum – platinum – rhodium thermocouple is connected by a flexible lead to the corresponding junction box. The installation is, in addition, fitted with a buzzer for signalling commencement and completion of the measurement.

The thermocouple wires (Fig. 3) were enclosed in a sheath taking the form of a stainless steel tube (or seamless steel tube) with dimensions 25 x 2.5 x 3000 mm.

The sheath carried a metal grid (10 x 10 x 1 mm) and was coated to a length of 2000 mm with refractory paste. The end of the sheath was protected with a fireclay or metal pocket 100 mm long. The working end of the thermocouple (length of projecting section not less than 50 mm) was protected by a quartz ferrule 75-80 mm long, outer diameter 7-9 mm and wall thickness 1.2 ± 0.3 mm).

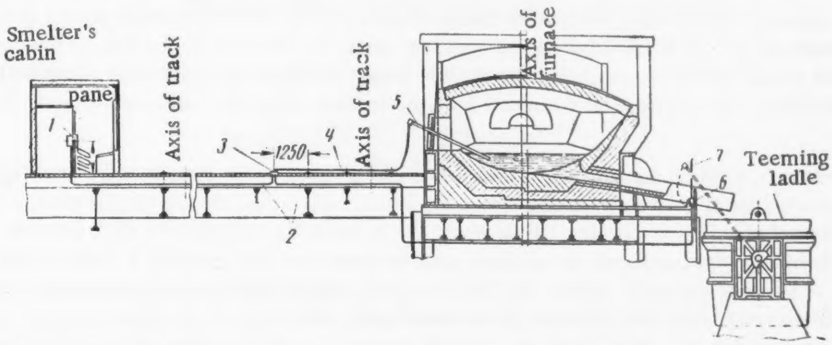

Fig. 1. Over-all arrangement for temperature measurement of liquid steel, immersion thermocouples:
1) valve potentiometer; 2) compensating lead-in tube; 3) No. 1 junction box; 4) compensating lead-in flexible metal sheath; 5) movable platinum-platinum-rhodium thermocouple; 6) No. 2 junction box; 7) screen against thermal radiation.

The thermocouple wires were insulated inside of the quartz ferrule by means of a porcelain tube, diameter 2× 0.7 mm. The ferrule was inserted into a steel plug which was screwed into the steel head, fixed in place with a pin. The joint was packed with calcined asbestos cord.

Two rods of 3 mm diameter steel wire were screwed into the head for rigidly fixing the thermocouple wires to the collars on the plates. The thermocouple wires were insulated with a double-bore porcelain tube diameter 7-9× 0.9-1.5 mm and on the remaining section with single-bore porcelain tubes, diameter 3.9× 0.9-1.5 mm, being wrapped with a double layer of glass tape. The free end of the thermocouple was connected to the compensating lead either by means of a plug connection or by soldering. The end of the compensating lead was led out through a textolite clamping block.

On account of deterioration of the insulation, the compensating lead had to be frequently changed in the tube. In the interests of economy and in order to employ the compensating lead already in use, a special reliable method of insulating it was developed. The cleaned lead was coated with a layer of silico-organic enamel 0.05 mm thick and baked in a furnace at 250°, more layers of glass thread being then applied to the wire with a silico-organic or gliftal (polythene) varnish and again treated in the furnace at 250°. Each lead was then coated with a glass sheath and the sheath was again impregnated and made up into a thin-walled metal tube with diameter 10-12 mm.

The construction of the thermocouple provides quick changing of the quartz ferrule, changing of the plug and other parts, quick dismantling and assembly and connecting up for the measurement. The thermocouple weighs 8-10 kg.

Temperature Test on the Metal in Open-hearth Furnaces

The metal temperature in the open-hearth furnaces is measured at the commencement of the final boiling, twenty minutes before reduction, before reduction and in the ladle. Before commencing the measurement, the instrument man connects in the made up thermocouple and checks the accuracy of the instrument. The steel smelter introduces the thermocouple into the furnace through the inspection hole of the middle charging vent, heats up to 1000° and on the first buzzer signal immerses the thermocouple in the metal at an angle of 45° to a depth of 150-250 mm. At 1200-1300° the recorder motor is switched in for a speed of 1 revolution every 3 minutes. After 12-15 seconds a steady temperature curve is observed on the chart. This signifies that the measurement is completed, and at the second buzzer the thermocouple is withdrawn from the furnace.

In order to measure the metal temperature in the ladle the panel and gear for immersing the thermocouple are set up. In order to protect the quartz ferrule from mechanical damage, a ram is fitted at the tip of the thermocouple. The entire process is repeated if the measurement is unsuccessful. The results of the measurement are recorded on the chart, on the smelt report sheet and in a special log book.

Measurement of the Metal Temperature in Electric Furnaces.

For the purpose of introducing immersion thermocouples into the electric furnace plant a number of investigations were carried out for establishing the optimum metal temperature in the bath of the electric furnace and ladle at all periods during the smelting process, viz. during the melt, on completion of the boiling, on commencing the refining, after drawing off the metal from the furnace, and in the ladle immediately after drawing off the smelt.

Since February, 1955, in the smelts involving all grades of steel in the electric steel smelting plant, temperature measurements have been carried out on the liquid steel during the smelting process at the commencement of the refining, before drawing off the smelt and in the ladle immediately after pouring. The change-over to steel smelting with systematic temperature measurements has made possible a considerable improvement in the quality of the smelted steel. Hence, in 1955 the reject of grade ShKh15 steel was reduced in the ratio of 2.4 : 1 and stainless steel in the ratio of 1.8 : 1 as compared with 1954.

Measurement of the steel temperature opened up the development and introduction of a new smelting technology for grade ShKh 15 steel. In the course of six months in 1955 spoilage on account of nonmetallic inclusions was completely eliminated.

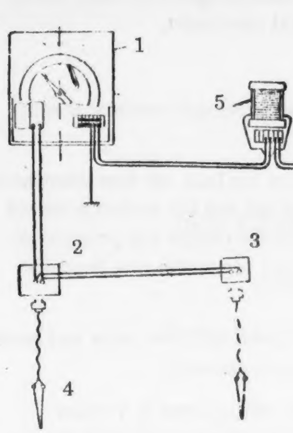

Fig. 2. External electrical connections of the valve potentiometer:
1) valve potentiometer EP-107; 2) No. 1 junction box; 3) No. 2 junction box; 4) movable thermocouple; 5) 220/127 v transformer.

Employing the optical pyrometer for the temperature measurement of stainless steel during tapping from the furnace required visual determination of the metal temperature during smelting by the foreman and steel smelter. This demanded a high level of experience and frequently resulted in an unsatisfactory chemical composition of the steel with reference to titanium, and even spoilage on account of incorrect metal temperature. The over-heated metal frequently welded onto the molds and required breaking up in order to extract the ingots.

The use of immersion thermocouples made it possible to regulate the metal temperature during smelting and to draw off the steel with only slight temperature variations. It was also possible to maintain the titanium content within narrow limits, to eliminate over-consumption of ferrotitanium, to increase the durability of the molds, to standardize heating of the slabs in the blooming pits, to improve the quality of the finished product, to raise the productivity of the electric furnaces in stainless-steel smelting and to achieve a saving in electrical energy.

The possibility of regulating the metal temperature during the smelting process provided an increase in the durability of the lining of the electric furnaces since over-heating of the bath was completely excluded.

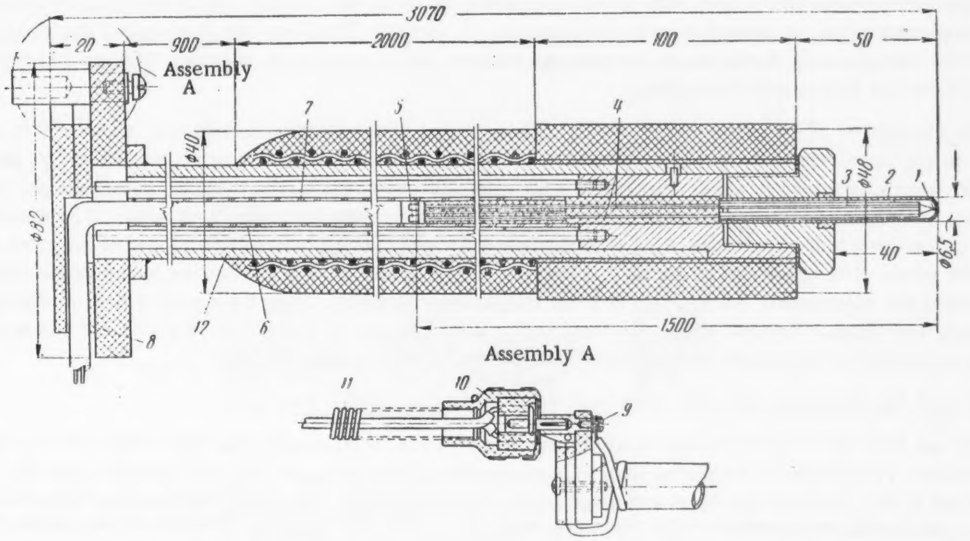

Fig. 3. Thermocouple for measuring the temperature of liquid steel:
1) hot junction of the thermocouple; 2) quartz ferrule; 3) beads, tube; 4) double-bore beads; 5) plug joints or soldering; 6) porcelain beads; 7) glass tape; 8) compensating lead; 9) textolite clamping block; 10) plug; 11) flexible lead; 12) seamless tube.

At the present time measurement of the metal temperature by means of thermocouples has been firmly established in practice, making it possible to completely dispense with the optical pyrometer.

Repair of Immersion Thermocouples.

The thermocouple is repaired when the characteristics vary, or when the thermocouple wires or sheath are damaged. Details of the repair are entered into a special log book.

Dismantling the thermocouple. In dismantling the thermocouple the ends of the lead are first disconnected from the junction box and the terminal block is removed, the plug being screwed out and the pocket removed. The thermocouple removed from the sheath together with the head are freed from the collars and attachment. The compensating lead is then disconnected and the glass tape, porcelain tubes and beads removed from the thermocouple.

The porcelain tubes and beads are treated with aqua regia for further use, rinsed with hot water and heated in a muffle furnace at 700-800°. The glass tape is drawn out and the wire and head cleaned.

Repair of the thermocouple wires. The thermocouple wires and working tip are restored to a sound condition by electric arc welding. The thermocouple wires must not be allowed to make contact with the electrodes of the arc. During welding the maximum current required by the electric arc should be 10-12 amp. at a voltage of 220 v.

The thermocouple is usually annealed and calibrated after 20-40 measurements, and sometimes after fewer measurements at the request of the foreman.

Assembling the thermocouple. In assembling the thermocouple,a porcelain tube, diameter 2× 0.7 mm is first placed on one leg of the thermocouple along the length of the quartz ferrule, followed by a double-bore porcelain tube. The remaining section is insulated with a porcelain tube or beads. The free end of the thermocouple is connected to the compensating lead and is insulated with glass tape. From the double-bore tube to the compensating lead the thermocouple carries a double layer of glass tape. The head carries stiffening wires, the thermocouple being fitted to it with clips, it is inserted in the sheath and the head is fixed in the clamp. The compensating lead is connected up to the terminals. A socket is fitted onto the front end of the sheath wound with asbestos cord, a plug is screwed into the head and the quartz ferrule is fitted. The insulation resistance must not be less than 2 megohms.

Reinforcement of the thermocouple sheath. The near end of the sheath is coated to a length of 2 m with heat-resisting paste, with a composition by volume of $^3/_4$ magnesite or chrome magnesite powder and $^1/_4$ ground fireclay carefully mixed to a stiff compositon with water glass (two parts by volume) and water (one part by volume). The sheath is lined in a special press. The surface of the press is covered with paper. The sheath is set up on the lower half of the press filled with the refractory paste, it is covered with a layer of the paste and then with paper. The upper part of the press is lowered, applying uniform compression by means of the clamping screws until the edges meet. After drying at room temperature for sixteen hours the sheath is fired at 200-300° for twenty-four hours. The fired sheath is coated with a lime solution. A metal grid 10 × 10 × 1.6 mm or 10 × 10 × 1 mm is applied to the sheath for purposes of strength before coating with paste.

Staff and Equipment of the Temperature Measurement Section

At the steel works approximately 5000 measurements of the metal temperature are carried out at present every month. Preparation of the thermocouples, maintenance of the equipment and the measurements are carried out by one foreman, one chief and fifteen shift electrical fitters. The chief electrical fitter supervises accurate and continuous operation of the measuring equipment. The shift electrical fitters repair and prepare the thermocouples, and carry out punctual and actual measurements of the metal temperature. The section carries log books for entering the results of temperature measurements and details of thermocouple repairs.

ROLLED STEEL AND TUBE MANUFACTURE

THE MANUFACTURE OF BENT SECTIONS

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Hot rolling does not always provide an accurate section. In this respect considerable advantages are provided by the method of rolling which makes it possible to obtain bent sections in the cold state from strip, bars and sheets on roll forming mills. The advantage of this method consists in the fact that the process of rolling the metal is distributed more favorably throughout the section. By this means it is possible to considerably reduce the weight of the sections for equal strength as compared with hot rolling. In addition, it is possible to obtain sections of varied form, thus raising the working characteristics of the section considerably.

Bent sections are widely employed in the building industry for cementless lamps, glass cases, window frames, doors and gutters; in the motor industry for windscreen frames, gauges, all types of edgings, and so on, (Fig. 1); in bicycle manufacture for rims and mudguards. Bent sections have more recently come into use in the metallurgical industry.

The method of forming yields a high level of productivity (up to 120 m/min) of the plant with simple equipment and a small number of workers. The quality of the surface of the sections is so high that the subsequent operations (polishing or decorative coating) can be carried out without further treatment.

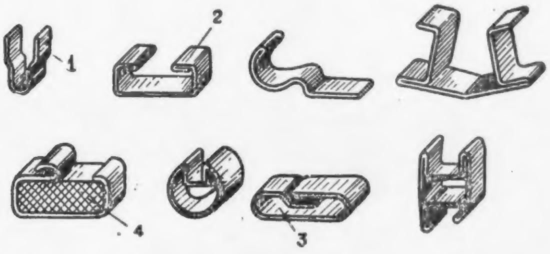

Fig. 1. Bent sections employed in the motor industry:
1) open type of section; 2) semi-enclosed; 3) enclosed; 4) packing.

The forming process can be combined with other technological processes such as welding (in the manufacture of welded tubes, Fig. 2) soldering (in the manufacture of radiator tubes) bending into an arc, punching, stamping, cutting to given lengths, and so on.

Cold forming differs in principle from hot rolling. While the thickness (height) is reduced in hot rolling, together with the cross section of the billet as a result of passing through the revolving rolls, in forming, only the shape of the cross section of the bar varies while the thickness and cross-sectional area remain the same.

The constant thickness of the material during forming likens this process to the process of bending bars in presses. The difference consists only in the fact that, first, during forming, the billet moves continuously forward and theoretically is of infinite length, while second, the forming tool does not move in steps but revolves.

Cold ruling is applied to steel strips and bars, nonferrous metals, stainless steel, and so on. The billet can be from 0.10 to 20 mm thick and up to 2000 mm wide. In Russian factories the coefficient of utilization of the metal is 99.5-99.7%.

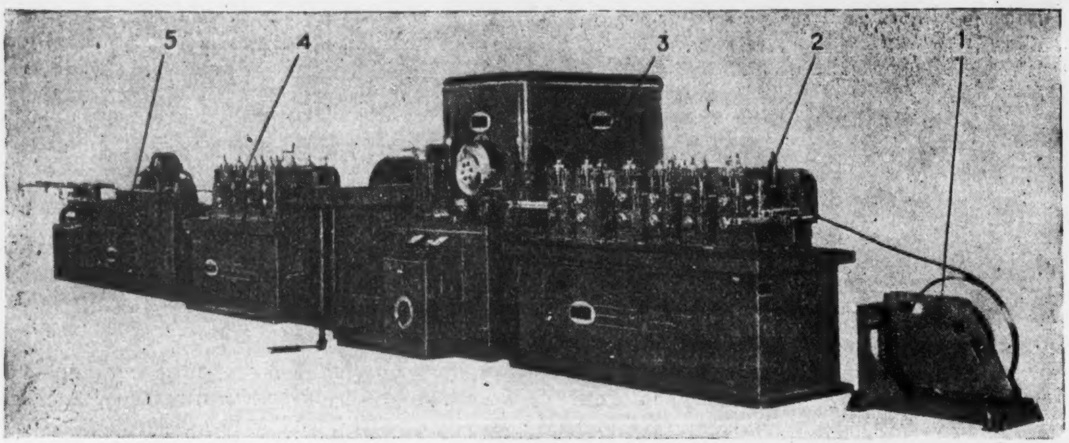

Fig. 2. Machine for manufacturing welded tubes:
1) reel; 2) shaping machine; 3) welding machine; 4) gauging machine; 5) tool turret.

Depending on the maximum thickness of the worked material, the cold rolling mills (Fig 2 and 3) taking the form of a number of rolling stands with a common drive, can be divided into three groups, namely light (thickness of billet up to 1.2 mm), medium (from 1.2 to 2.5 mm), and heavy (over 2.5 mm).

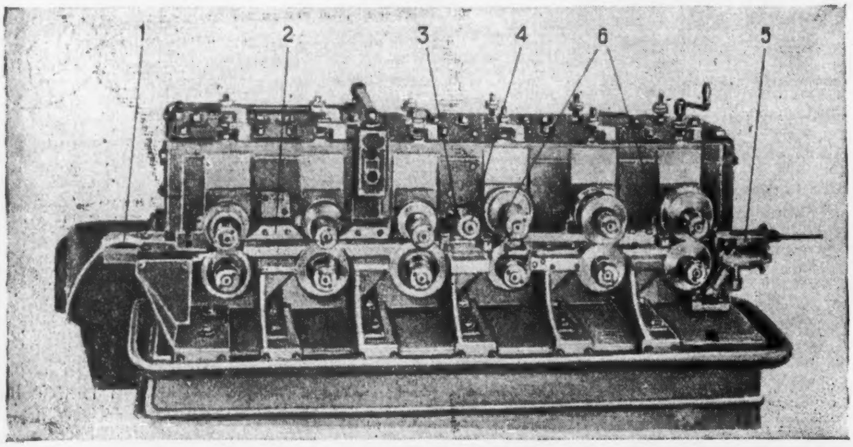

Fig. 3. Light cold-rolling mill:
1) entry guides; 2) intermediate guides; 3) bracket; 4) top auxiliary roll; 5) take-off guides; 6) end auxiliary rolls.

On light mills (Fig. 3) the shafts carrying the working rolls operate as cantilever beams. This represents a disadvantage, since in the case of deformation under loading, the axes of the upper and lower shafts cease to be parallel.

On medium and heavy mills (Fig. 2) the shafts operate as beams resting on two supports, while the ends of the shafts revolve in special detachable chocks. The top shafts are controlled vertically, by means of which it is possible to vary the tractive force (compressive force on the blank). The medium mills have a maximum number of pairs of spindles (up to 24) while the light type of mill usually has not more than 8 pairs. The heavy mills do not differ in construction from medium mills but the shafts are considerably thicker (up to 375 mm, and and on medium mills 50-60 mm).

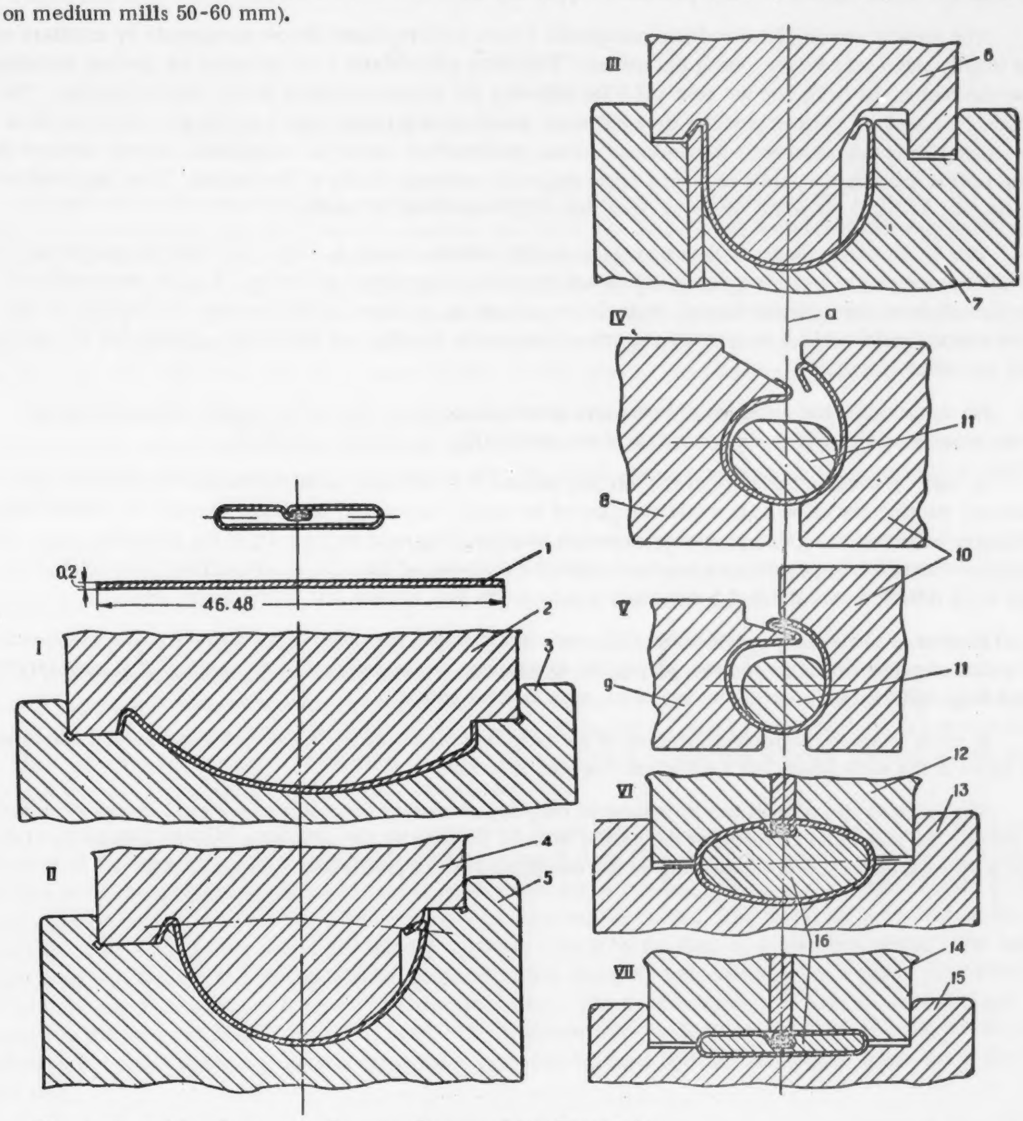

Fig. 4. Transitional stages in the manufacture of a radiator tube :
1) blank ; 2,4, 6, 12 and 14) top rolls ; 3,5,7,13 and 15) bottom rolls ; 8,9) end auxiliary rolls ;
10) intermediate forming guides ; 11,16) mandrels, fillers.

The main forming tools (the active rolls) are made from alloyed steel or cast-iron (for a large section). In the manufacture of simple sections from thin sheet and in small batches (50-75 thousand m), plasticized timber rolls can be employed. Steel rolls have a high degree of strength; without polishing the rolls, more than 1 million m of section can be manufactured from mild steel and with polishing together with appropriate treatment up to 15 million m and over. The accuracy of dimensions of the rolls is ± 0.05 mm, and the working surfaces must be polished.

The auxiliary end rolls 6 (Fig.3) rotate around a vertical axis; the axis of rotation of the upper auxiliary rolls 4 can be set at any required angle to the horizontal, depending on the design of the bracket 3 to which the rolls are fixed. In this way it is possible to apply any desired bend to the object.

The rapidly wearing intermediate guide plates 2 have been replaced almost completely by auxiliary rolls and usually serve only for supporting the section. The entry guide plates 1 are intended for guiding the billet from the reel to the mill, and the take-off 5 for relieving the stresses occurring in the finished section. The length of the take-off plates is not less than 250 mm. Mandrels and fillers (Figs. 1 and 4) are employed when it is necessary to obtain an exactly dimensioned section profile which cannot be compressed directly between the main rolls or the auxiliary rolls or when a rigid support is necessary inside of the section. They are fixed to the bracket 3 (Fig. 3) at points where an open form still remains in the profile.

The intermediate shapes in developing the profile must be chosen in such a way that the process is carried out without temporary secondary deformations and excessive expenditure of energy. It is not permissible to successively bend the material through an angle, to reshape it, to allow friction between the surfaces of the blank and to bend the blank in opposite directions (successive bending and unbending), giving rise to hollows, folds and other defects.

For determining the sequence of successive intermediate shapes the rule is usually followed that the deformation should pass from the main axis of the profile (Fig. 4, a) to the periphery.

In order to obtain continuity of the forming process it is necessary to set up on each pair of rolls the necessary traction forces, so as to avoid dragging of the strip (longitudinal bending). Hence, the rolls of each successive pair must have a mean speed of rotation somehwat higher than the rolls of the preceding pair. Experience at the Molotov car factory has shown that the diameter of the rolls should increase successively by 0.4% when rolling material 0.3-2.5 mm thick and by 0.25% for material less than 0.3 mm thick.

In order to obviate the considerable difference in the linear velocities of the various points of the rolls at the points of contact with the section, giving rise to scratches on the surfaces of the section, it is necessary to avoid large differences in the diameters of the rolls at these points.

In order to avoid longitudinal bending of the strip, it is necessary to choose such intermediate shapes that the billet in the early passes has a sufficient moment of inertia.

The required size of mill can be estimated only approximately according to the total energy expended in bending the individual elements of the profile section, knowing at the same time that the efficiency of the mill is approximately 0.6-0.4 depending on the complexity of the profile and the type of mill.

VARIABLE-DIAMETER TUBE DRAWING

M. A. Freiberg

Manager, Tube Drawing Plant of the First Ural New Tube Factory

Variable-diameter seamless tubes are widely employed in machine construction. For example, the trolley-bus collector bar takes the form of a seamless tube 5800 mm long in four sections of different diameter (51, 44, 35 and 25.5 mm). The large-diameter tube is connected at the end to the trolley bus. Since the greatest bending moment occurs at this point, the tube must have the greatest rigidity here. The free end of the tube carrying the collector shoe must be very flexible and therefore has a smaller diameter. Tubes are employed in a motorcycle frame having not only a variable diameter along the length, but also a variable cross section. One end of the tube has a round section and the other an oval section (Fig. 1).

Constant-diameter tubes are usually manufactured by drawing the entire tube through the draw-hole. In the manufacture of variable-diameter tubes, only part of the tube is drawn. This is achieved by means of a special device limiting the movement of the bogey (Fig. 2), which allows tube drawing to cease without stopping the machine.

Variable-diameter tube drawing is carried out on a 30-ton drawing machine with a speed of movement of the working chain of 15 m/min. In order to convert to variable-diameter tube drawing there is an auxiliary bogey with a special hook which on connecting draws for not more than five minutes.

The draw ring is fixed in the collar plate with bolts by means of the bearing plate. The bolts hold the ring in the collar plate also during return of the drawn section of the tube. The tube is returned by means of the bogey which is connected to a reverse-drive chain. Moving up towards the collar plate, the bogey seizes the head of the tube and draws it backwards to the ring. When the working collar of the draw-hole is 3-4 mm wide the diameter of the drawn section of the tube is approximately 0.07-0.11 mm less than the diameter of the draw-hold and the tube is readily drawn backwards through the ring.

As the working collar increases, the diameter of the drawn section of the tube becomes equal to thebogey diameter of the draw-hole and the force of the bogey is insufficient for drawing the tube in the ring. The ring has then to be changed and the jammed tube may be drawn out of the ring by means of a special device (Fig. 3), consisting of a hollow cone cut into four segments with transverse incisions. The segments are seated on the cone belt at an angle to the axis of 10° and are tightened onto it by the rings of a wound-on spring. The end of the cone bolt finishes in a shoulder which connects with a clamping fork connected to a cable. The other end of the cable is attached to the bar of a pneumatic drive. The cone bolt, together with the segments, are inserted in the tube and pulled slightly by hand. The segments with the incisions then tightly seize the inner surface of the tube. Reversing, the bar draws the cable with the cone bolt, and the tube is drawn out of the draw ring.

The technology and gauging of variable-diameter tube drawing can be learned on the basis of an example, namely manufacture of collector rods.

Collector rods are manufactured from grade 30KhGSA, steel drawn in four passes. The initial blank consists of a hot-rolled tube 57 × 3.5 × 4800 mm. Before drawing, the blank is pickled in a 20% sulfuric acid solution for removing grease. Blanks on which pickling displays effects (hair-line cracks, cavities, blow holes) are removed from the batch.

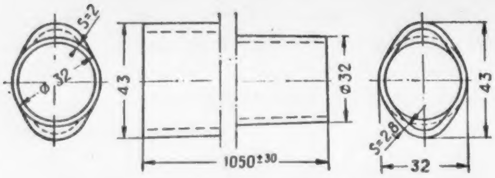

Fig. 1. Large fork for motorcycle frame.

Suitable blanks are put to the pneumatic hammer for stamping the heads. The ends of the tubes before stamping are heated in a forge up to 780-850°. Heating has to be observed very strictly since over-heating produces cracking of the heads and under-heating leads to flaking. The length of the stamped heads must be uniform for all the tubes, namely 200 mm.

After stamping the heads, the batch of heads is annealed in batch furnaces at 680-700°, being held at this temperature for 1 hour 20 minutes and then cooled in air.

During annealing, the tubes bend slightly and must therefore be straightened on a cogging straightening machine, and returned to the batch, dividing the horizontal layers of tubes with covers, to be pickled again, rinsed and phosphatized.

After washing in a soap emulsion solution, the tubes are passed to the drawing machine. The first pass is on the short straightener on the system: 57 × 3.5 mm → 51 × 3.2 mm. Heads are driven on the drawn out tubes, preliminary heating is applied in the forge and then the tubes are again annealed, straightened, passed through the pickling baths and subjected to further drawing.

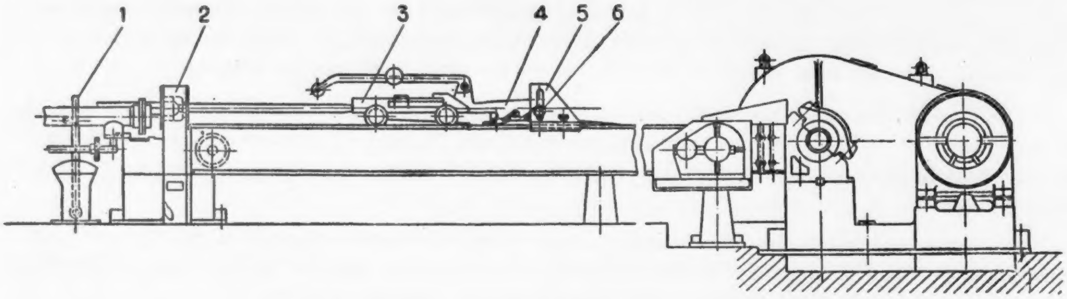

Fig. 2. Plant with bogey stop for wire drawing: 1) rod-shift pneumatic drive control; 2) collar plate; 3) bogey; 4) grapple with tapered face; 5) bearing plate for bogey stop; 6) roller for disengaging grapple from the driving chain.

Since the second, third and fourth passes are without mandrels (pressure drawn) the initial annealing of the tubes is carried out on another system: heating to 780-800°, holding for fifteen minutes and cooling in air. On interrupting this system in the process of mandrel-less tube drawing the tubes may crack along the generator.

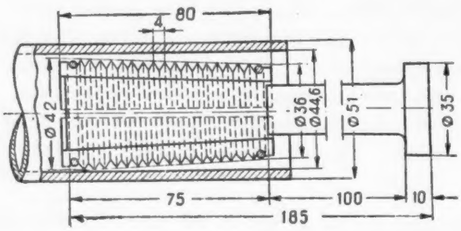

Fig. 3. Device for extracting the tube.

The second pass is given to only part of the tube through a draw-hole with diameter 44.1 mm. Since drawing is carried out under pressure and the wall thickness is increased after each pass by approximately 0.1 mm the drawn part of the tube has a wall thickness 3.3 mm after the second pass, 3.4 mm after the third pass, and 3.5 mm after the fourth pass.

Hence, on the large-diameter end, the wall thickness of the tube is 3.2 mm and on the small diameter end 3.5 mm.

Preparation of the tubes for the third pass is carried out in the same way as for the first two, except that for heating the tubes all the burners are not opened up, but only those located along the path of the batch of tubes at the end carrying the heads, hence, thermal treatment is applied only on the part of the tube where hardening occurs.

The third pass is through a draw-hole with diameter 35.1 mm and the bogey stop is moved nearer to the collar plate.

After this pass the heads of the tubes are cut off. The tubes are first annealed in order to remove the hardness and to prevent the formation of cracks as a consequence of dynamic impact of the cutting wheel.

Cutting off the heads of the tubes having three knees after the third pass is a difficult operation, as a result of which it is necessary to obtain a completely identical length of all the tubes after cutting them to the diameter 35 mm. If this is not observed, the length of the third section of the tubes after the fourth pass will be different. Calculation of the length in cutting the tubes is carried out from the joint with diameter 44 and 35 mm.

New heads are driven on the pneumatic hammer, the ends of the tubes being previously heated. The length of the heads on all the tubes must also be exactly the same (180 mm).

The fourth final pass is made through a draw-hole with diameter 25.6 mm.

Setting the bogey stop is a most responsible operation in drawing variable-diameter tubes since the distance of the stop from the collar plate governs the length not only of the drawn section but also of the section remaining at the larger diameter. This distance must be calculated.

The finished tubes are subjected to thermal treatment—annealing at 400-500° and normalizing at 850-860° (heating thirty minutes, holding fifteen minutes, cooling in air). For quick uniform cooling during normalization the tubes are set up on hoops in a single row.

After thermal treatment the tubes are sent for finishing.

The normalized tubes are first straightened on the straightening machine and finally reeled.

In the rolling mill the tube is given a larger diameter and as it moves forward with rapid rotation of the control wheel, the rolls approach up to contact with surface of the successive section of the tube. For straightening one tube it is necessary to bring the rolls together four times before being opened up for the next tube. The speed of tube straightening is 0.6 m/sec.

After straightening, the ends of the tubes are cut off and samples are taken for mechanical testing. If the ultimate strength of the tube is not less than 70 kg/mm^2 and elongation of a sample 1/10th of the length is not less than 11%, then the tubes are considered satisfactory.

Variable-diameter seamless tubes are a fully established grade and can be manufactured on standard equipment of the tube drawing plant.

IMPROVING THE QUALITY OF HOLLOW DRILL STEEL

Hollow drill steel rods are employed in the mining industry for drilling the blast holes in the rock. During operation the rods are subjected to impact loads which give rise to considerable alternating stresses in the metal. In addition, the rods are subjected to the corrosive effect of the water flowing through the channel. A high-quality metal surface and accurate geometrical dimensions of the rods are therefore necessary.

The following articles provide manufacturing experience with hollow drill steel at various metallurgical plants.

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Hammer and Sickle Plant

For rolling hollow drill steel a U7 carbon steel billet is employed with a square section, having a hole through the center taking an EI94 austenite steel core.

One of the main features of rolling this type of billet from the point of view of deformation of the metal is the appreciable difference in the strengths of the carbon and the austenite steels. With this difference in strength simultaneous deformation of the core (austenite steel) and the sheath (U7 steel) during hot rolling would be difficult to achieve if there were an insufficient force of friction between them in the process of deformation.

The bond between the core and the sheath, governed by this force of friction, strives to maintain equilibrium between the two elongations, that is to say, to reduce the elongation of the sheath and to increase the elongation of the core. In this way retardation of elongation of the sheath and the two-way pull of the core approximate their deformation resistance.

Another typical feature of rolling the billet is the absence of direct contact of the core with the rolls; deformation of the core is applied by the metal of the sheath. Hence, the nature of the core deformation is governed entirely by the quantity and directions of the displacements of the metal sheath at the seat of deformation of the given pass.

Large transverse displacements of the metal sheath at the seat of deformation are undesirable, since very slight asymmetry relative to the core gives rise to dislocation of distortion. It is very difficult, again, to achieve full symmetry of the transverse displacements of the metal in practice; to obtain this it would be necessary to provide completely uniform heating of the billet throughout the section, together with exact positioning of the rolls during reduction. It is therefore necessary during the rolling to limit the value of the relative reductions in the passes and to employ a system of passes for which sharp transverse displacements of the metal of the sheath do not occur at the seat of deformation.

An important part of rolling drill steel is the system of drawn rhomboid and square passes, in which there is the greatest possibility of sharp nonuniformities occurring in the distribution of the relative reductions at the seat of deformation.

Greatest reliability is afforded by rolling in a series of rhomboid and square passes with marked gaps in each pass. A disadvantage of this method is the necessity for limiting the reduction per pass; that is to say, for increasing the number of passes or reducing the cross-section of the billet.

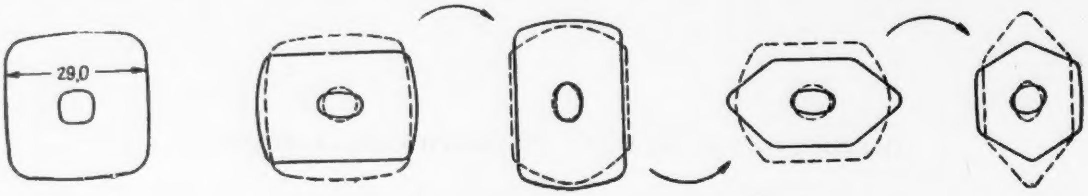

Fig. 1. Irrational shape of finsihing passes (rolling 22 mm hexagogonal).

In the roughing passes the distribution of the relative reductions at the seat of deformation does not give rise to such sharp forming of the core by the sheath metal as in the case of the rhomboid. This phenomenon may occur at the seat of deformation of the finishing pass when an unsuitable pre-finishing pass profile is employed (Fig. 1); this can be obviated by choosing a pre-finishing pass with sharp angles (Fig. 2) which provides uniform distribution of the relative reductions at the seat of deformation in the finishing pass.

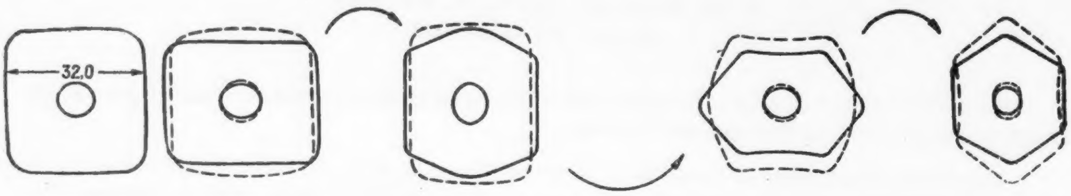

Fig. 2. Rational shape of finishing passes (rolling 25 mm hexagonal).

After a series of investigations the technology of rolling hollow drill steel was considerably improved at the Hammer and Sickle plant:

1) heating of the billet was improved by increasing the period in the holding furnace of the 300 mill from 1 hour 15 minutes to 1 hour 45 minutes and raising the initial rolling temperature from 1070 to 1100°;

2) the number of draw passes was increased from 7 to 9 in a number of rhomboid passes of the reducing stand by employing passes adjacent to the third and fifth passes. This was done for the purpose of avoiding overfilling of the passes, the over-all number of passes being increased from 14 to 16 for rolling 25 mm hexagon sections and from 16 to 18 for rolling 22 mm hexagon sections;

3) a pre-finishing profile was chosen providing uniform distribution of the relative reductions at the seat of deformation of the finishing pass, as a result of which no gap was formed between the core and the sheath;

4) a periodic check on the shape and position of the core in the rolled sections was introduced, occurring in the roughing, pre-finsihing and finishing passes. The check was carried out during the rolling process by cutting templets with a thin emery wheel. The templets from the roughing square sections (12th and 14th passes) provided an assessment of the quality of the setting in the draw pass system.

A disadvantage of the drill-steel rolling technology at the Hammer and Sickle plant was the considerable displacement of the core (hole) in the initial billet on account of "drift" of the drill during drilling and also as a result of bending of the billet.

The permissible variation in the sheath at the faces after drilling is not more than 6 mm and the permissible bending not more than 3 mm/lineal m. Obliquity of the billet face must not exceed 10 mm.

These tolerances cannot be regarded as sufficiently strict. It was soon found necessary to carry out straightening of the billet for the purpose of reducing the maximum curvature to 1.5 mm.

The billet must be well and uniformly heated. Nonuniform heating with a given setting of the mill produces an incorrectly shaped channel displaced from the axis of the billets.

Over-filling of the rolls in the first stand frequently leads to displacement and distortion of the core and is also the cause of slipping of the sheath from the core. This considerably complicates setting of the roughing, pre-finishing and finishing passes.

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V. D. Senkov — Rolling mill foreman, Serov plant

During the last few years work has been carried out at the plant for improving the manufacturing technology of drill steel. The work was aimed at increasing the weight of the billet, varying the arrangement of the passes in the rolls of the reducing stand, increasing the length and improving the appearance of the finished product.

As a result of this work the percentage of standard product as compared with 1951 rose from 82.4 to 85.8% in 1955. The reject figure fell by a factor of 2.5.

For a long time, drill steel was rolled from a 115×115×900 mm billet weighing 80 kg. The billet arrived at the 850 mill after a large number of passes, which complicated the work of the large-batch factory. In order to raise the productivity of the rolling mill it was necessary to increase the weight of the billet.

To this end a 10-ton electric bridge crane was first erected on a new site together with three electric telpher lines each with a capacity of 0.25 tons.

A number of modifications were carried out in the design of the drilling machines: the spindles were fitted with specially prepared chucks at the rear stocks and self-centering chucks at the first stocks. This eliminated displacement of the holes in drilling and expedited fixing of the billet.

These measures made it possible to carry out drilling on a billet with dimensions of 125 × 125 × 1000 mm, as a result of which the productivity of the mills was raised by 16%.

On increasing the size of the billet it was necessary also to increase the diameter of the hole.

Calculation of the hole in a 125 × 125 mm billet was carried out according to the formula

$$d_1 = d_0 \sqrt{\lambda},$$

where d_1 is the diameter of the hole in the billet, mm;
d_0 is the diameter of the hole in the finished section, mm;
λ is the total reduction of the metal.

It was established by calculation that for a 125 × 125 mm billet the diameter of the hole must be within the range 38-40 mm. The accuracy of the calculation was confirmed by rolling an experimental batch of billets with a hole diameter 38.3 mm to 22 and 25 mm hexagon sections and 32 mm rounds.

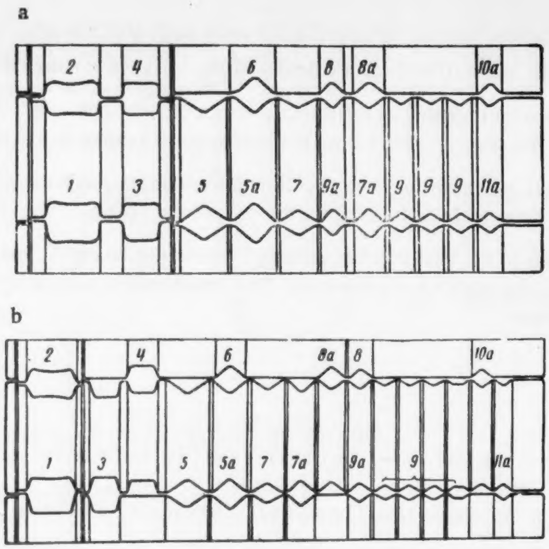

Fig. 1. Systems of passes in sinking rolls for rolling batch and drill steel:
a) old; b) new.

The high demands on shape and position of the hole in the finished section require an accurately worked-out setting of the rolls on the reducing and finishing passes.

Great difficulties are encountered in designing the rolls of the reducing stand, since according to the conditions of manufacture it is impossible to have an individual set of rolls for the reducing stand for rolling drill steel and therefore the reducing rolls must be set for simultaneous rolling of drill and light-alloy batch steel (Fig. 1, a).

In this case the batch steel is rolled in pases 1-9 and drill steel in passes 1,2,3,4,5a,6,7a,8a,9a,10a and 11a. Pases 1,2,3,4, and 6 are common. This is the old arrangement of passes. Its disadvantage is the large number of passes adjacent to each other and consequently it is necessary to roll the billet in rolls with widely different working diameters. This leads to distortion and displacement of the hole in the finished product.

A new pass arrangement has now been worked out for the rolls of the reducing stand. The length of the rolls was increased by 200 mm and some of the previously adjacent passes were arranged in zig-zag order (Fig. 1,b).

Experimental rolling of a billet with the new arrangement of passes in the rolls of the reducing stand showed satisfactory results.

In rolling hexagon sections the most common systems have the roll passes of the finishing train such that either the middle of the side of the section or the angle of the hexagon occur at the points of separation of the finishing passes.

Analysis of the elongation charts together with experience obtained indicate that the second system of passes in the rolling of drill steel yields better results. In operating this system, however, difficulties were encountered which could not be overcome at the plant. The bar twisted even when operating with forming wires and a small gap between the bar and the wire. The problem of setting the bar in the wires with minimum gap is difficult, leading to an increase in the idle time and substantial cooling of the metal.

Arising from this, rolling of hexagon sections in the finishing-mill passes is carried out at this plant according to the system shown in Fig. 2.

It was also established from the investigations that the following conditions have to be observed in order to obtain the correct shape of hole in the finished product:

1) correct shape of the finishing square section;
2) constant heating temperature of the rod and billet as well as degree of reduction and extension;
3) stability of the bar when passing through the roll;
4) difference between the diameters of the rod and hole in the billet not greater than 1 mm.

On changing over to rolling drill steel from a 125 × 125 mm billet the number of passes in the reducing stand and in the finishing train does not change.

By raising the weight of the billet from 80 to 110 kg it was possible to raise the productivity of the 320 mill when rolling drill steel by 11.4%.

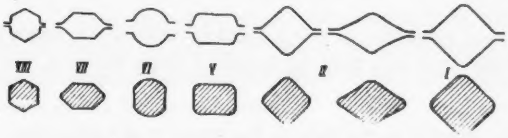

Fig. 2. System of rolling in the finishing tain for 22 mm hexagonal drill-steel.

Work was also undertaken for improving the quality of the finished product. In accordance with the requirements of the user, the length of drill steel was increased and, since the length of the rod was limited by the length of the draw bench on which the rod was drawn, the length of the draw bench was increased accordingly. At present, part of the drill steel is ordered by the users to a length up to 6.5 m.

Tests on the quality of the drills carried out by the users showed that several cases of drill breakage have occurred during operation recently. Since in the majority of cases a pobedit, tungsten carbide-titanium carbide alloy tip is welded onto the end of the drill and the drill steel is employed only in the capacity of a holder, the question arose as to the suitability of employing grade U7 steel, without annealing, for manufacturing the drill steel.

Tests were carried out on other grades of steel. For this purpose three test batches of drill steel of the 25 mm size were prepared from U7 steel (without annealing, and annealed) and grade 45 steel.

The finished drills were tested on the south Ural bauxite mines. It was found that the toughest drills were those manufactured from U7 steel with annealing (40.3 blast hole-meters to 1 drill failure).

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V. V. Turitsyn – Central Factory laboratory steel sections group manager,

Red October factory

Hollow hexagon drill steel section manufactured from grades U7 and U8 steel is widely employed in the mining industry. According to the specification TU 2894-51 the internal diameter of the hexagon section must be in mm:

For the 22 mm hexagon section	5.8-7.7
For the 25 mm hexagon section	6.4-8.4

The core necessary for providing the hole in the hexagon section is prepared from grade EI94 steel.

The hexagon drill steel section is rolled on the 325 mill at the Red October factory.

In order to provide the correct dimensions and shape of the hole there must be a minimum gap between the core and the initial billet. In addition, uniform heating of the core throughout the section and along the length must be applied together with checking of the dimensions of the hole at the intermediate stands of the mill during rolling.

The old rolling technology for hexagonal drill-steel sections with the core inserted in the hot state did not justify itself since the correct shape of the hole could not be obtained on account of the sharp fluctuation in the heating temperature of the core and billet (instead of a round hole an oval or square hole was obtained). In addition, the rods frequently fractured on drawing out the core from the drill-steel sheath.

Investigations were carried out for improving the technology of manufacturing the hexagonal section.

The initial billet for rolling took the form of a 90 mm square section with a 27 mm diameter hole. The diameter of the core was 26 mm. The core was immersed in a milk-of-lime solution and then inserted in the cold state in the hole in the billet before setting in the furnace. The billet was heated for 3-5 hours to a temperature of 1100-1050° before rolling.

In rolling the first experimental batches of the drill steel according to this technology with cold insertion of the core, it was found that the core previously used, being 1000-1050 mm long for length of billet 900 mm, was excessively long.

In the first experiments a core of the same length was inserted in the hole in the billet in the cold state in such a way that it projected by 100-150 mm at the back end and was level with the billet at the front end. The projecting end of the core was bent slightly with a hammer before entering the first pass.

In the first two passes the billet over-ran the core, but did not completely cover it. At the third pass the core was pinched and the billet did not further over-run. The rear exposed end of the core frequently broke off, which gave rise to untimely wear of the surface of the finishing roll.

It is therefore necessary to have a core of limited length so that it is completely covered by the billet in the first passes in the reducing stand. At the same time the core temperature is held constant and is deformed together with the billet.

In order to determine the optimum core length seven experimental billets were rolled (90 mm square section) with cores up to 950 mm long inserted in the cold state. After rolling the billets to 25 mm hexagonal sections, samples 1 m long were cut from both ends of each bar, and washers were cut at every 100 mm for checking the diameter of the hole in the hexagonal section.

Analysis of the samples showed that the neck of the core was drawn out at both ends of each bar to 400-700 mm. In the remainder, the diameter and configuration of the hole were normal throughout the entire length of the bar.

It was established on the basis of the experiments that the optimum core length at which the billet completely covers the rear end must be 930-940 mm. In this case deformation in the finishing rolls will proceed without fracture of the end. Smaller cores would result in voids at the ends of the hexagonal section and would seriously distort the shape of the hole.

For stability in rolling 22 mm hexagonal sections it was necessary to slightly increase the diameter of the hole in the billet.

It was established by calculation that a 6.35 mm diameter hole could be obtained in the finished 22 mm hexagonal section (minimum according to specification TU 5.8 mm). With this diameter several difficulties in rolling were removed.

Accuracy of shape of the hole and limitation of core breakages in drawing out also depend on the process of drilling the billet on the horizontal drilling machines. The absence of collar plates (taking up the thrusts) on the machines gives rise to heavy vibration of the drill itself, resulting in a nonuniform hole-diameter along the length of the billet and sometimes to displacement of the hole from the center.

An axial section of a drilled billet demonstrated that on using a drill which had been previously used (diameter approximately 27 mm), the front ends of the billets had a hole diameter 27.3-27.8 mm and the back ends (at the end where the drill leaves the billet) 26.9-27.5 mm. In addition it was established that the internal diameter in the billet at a distance $^1/_3$ of the length from the end where the drill project, had a minimum size of 26.7 mm.

Hence, the diameter of the core inserted into the billet hole must be slightly less than 26.7 mm. Otherwise, the core seizes. To check the minimum diameter a calibrated rod with diameter 26 mm + 0.2 mm was inserted in the billet hole.

In order to reduce vibration of the drill and to obtain a hole uniform throughout the length of the billet, collar plates were set up on the horizontal drilling machine (taking the thrusts) for guiding the drill. This considerably improved the quality of the rolled hexagonal drill-steel section.

The minimum diameter of the hole in the billet when employing a drill previously used has to be measured by means of a reference core; the gap between the billet and the reference core must not exceed 1.0 mm.

As a result of the experiments the drilling technology was improved, the core length was established for the 90 mm square billet, the optimum heating temperature was found for the drill-steel billet (1100-1050°), and the correct diameter and shape of the hole in the hexagonal section were found, conforming to the requirements of the TU specification.

On the basis of these investigations a new technology was developed and introduced for the manufacture of hexagonal drill-steel sections.

On this basis it was possible to raise the productivity of the 325 mill by 15-20% and to turn out hollow hexagonal drill-steel sections in complete conformity to the requirements of the TU technical specification.

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One of the conditions for obtaining high-strength drill rods is accuracy of the geometrical dimensions of the cross sections.

It is particularly important that the channel in the rod should have a geometrically-circular cross section, since only under these conditions is a uniform distribution of the stresses throughout the whole cross section of the profile achieved. If the channel does not have a true round section and particularly if the axis of the channel is displaced with respect to the axis of the rod, then at points where the active section is reduced, excessive stresses occur giving rise to untimely wear of the rod.

In recent years hollow drill-steel has been rolled in practically the same way as ordinary hexagonal steel (disregarding specific characteristics), employing the same setting of the rolls. The quality was checked only after rolling in the last stand. Adjustment of the profile for correct shape of the channel was not carried out.

In recent years the quality of hollow drill-steel has been considerably improved. The changeover to simultaneous heating of the billet and insertion of the core in the cold state was first introduced at the Hammer and Sickle plant. This provided a substantial improvement in the shape of the channel. The setting of the profile according to the shape of the channel was first organized at the Red October plant, for this purpose setting up the emery cutting wheel directly near the mill. At all plants heating of the billet before rolling was improved, providing uniform heating of the metal. A substantial improvement was brought about in drilling the billet and so on. At the same time, insufficient attention was turned to setting the rolls.

The investigations carried out by the Central Iron and Steel Scientific Research Institute in conjunction with the workers of the Hammer and Sickle plant indicated a predominant influence on the setting of the rolls on the quality of the hollow drill-steel, particularly with regard to the rolls of the roughing and finishing stands. In the course of the investigations an attempt was made to obtain a hollow drill-steel corresponding to the best samples in regard to the quality of the section. The size tolerances were reduced to less than half: oval shape of the channel from plus 1.2-minus 0.8 to plus 0.5-minus 0.3 mm, that is to say the over-all oval tolerance of the channel was reduced from 2.0 to 0.8 mm and displacement of the center from 1.5 to 0.75 mm.

The investigations showed that hollow drill-steel with these tolerances could be obtained by introducing essential changes in the roll setting. Uniform deformation of the metal must be provided in each pass and conditions set up which would render difficult or reduce the tendency for the metal of the billet (sheath) to leave the core. For this purpose it was necessary to employ a rhomboid groove as far as possible with similar angles at the top. The finishing square section must have equal diagonals and this can be achieved by double rolling of the bar in similar grooves. The core in the finishing square section must be round and occupy the center of the cross section. The last three or four shaping grooves in which the profile is shaped are particularly important.

The investigations also showed that the best results are obtained on rolling in the grooves illustrated in Fig. 1. The conditions of deformation in these grooves are so favorable that on rolling in these grooves it is possible to obtain a hollow drill steel of almost ideal quality not only from the middle but from the front and back ends of the bar section also.

This setting has not so far been adopted in our factory. It requires a higher level of production, more careful setting of the profile, a better condition of the roughing and finishing stands, higher quality fittings and so on.

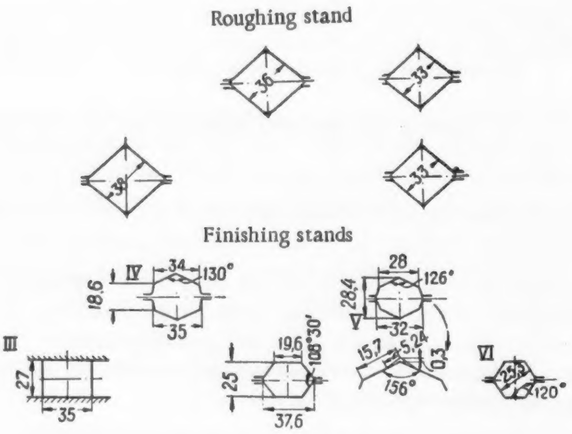
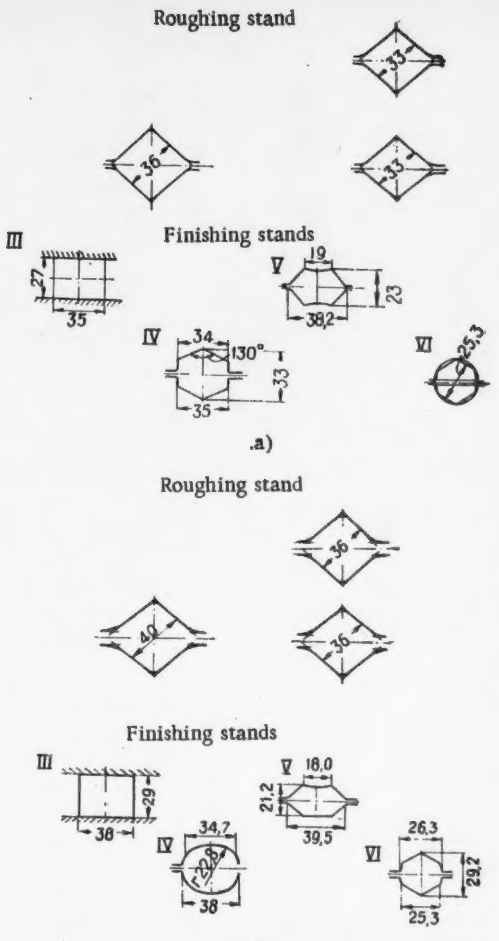


Fig. 1. Adjustment of rolls on the 300 mill for rolling high-quality hollow drill steel: III, IV, etc Number of stand.

The quality of hollow drill-steel was considerably improved at the Hammer and Sickle plant after introducing the pass sequence as recommended by the Central Iron and Steel Scientific Research Institute (Fig. 2, a) in place of that earlier employed (Fig. 2, b). One important feature of this sequence is that pressing out is considerably reduced particularly in the finishing pass. Instead of the finishing square section of 36 mm, a finishing square section of 32 mm was introduced, thus reducing the total elongation in the four passes from 2.7 to 2.15; the round upset pass was changed for a hexagonal and the pre-finishing pass was substantially modified: the width (height) was reduced by increasing the slope of the walls. In order to carry out better the finishing pass, the thickness of the pre-finishing pass was increased.



b)

Fig. 2. Setting of rolls for rolling hollow drill-steel: a) improved; b) old; III, IV, etc.) number of stand.

Practice showed that rolling in the finishing pass provided conditions for which the metal of the sheath did not leave the core and the channel was not deflected following reduction in the squeezing and broadening process.

The investigations also showed that in order to obtain a high-quality hollow drill-steel it was necessary to apply more rigid specifications with regard to the preparation of the billet for rolling. The billet had to be straightened before drilling; the deflection of the billet must not exceed 1.0 mm; the center displacement of the hole in the raw billet during drilling must not exceed 0.75 mm.

It is important to provide a hollow drill-steel with a good surface. This is provided by cleaning the billet. In order to obtain a channel with a good quality surface it is necessary to apply a new method for removing the core from the rolled rods. At the present time, in all the Russian factories, the cores are removed by the extraction method. This method is as follows. The cores are drawn out through the rod on chain draw-machines. By reason of the fact that the cross section of the back end of the core is scarcely diminished, on drawing out, along the inner surface of the channel there are formed scratches and score marks and the metal adheres. This all has an injurious effect on the quality of the channel surface, since in the operation of the rods the stresses are concentrated at the faulty locations and fracture occurs.

The method of preliminary stretching is a much more advanced method for removing the core. The core is initially stretched at both ends. By reason of the characteristics of the austenite steel from which the core is made, it diminishes uniformly in cross section along the length during elongation without forming necks. As this proceeds the metal of the core leaves the billet (sheath). After preliminary elongation by 20-30% the core readily leaves the rolled bar. The channel surface is clean and without defects.

In the course of experiment it was established that after preliminary stretching of the core by 25% it can readily be extracted from a bar 15-17 m long, even manually.

Removal of the core by the method of preliminary stretching provides not only an improved quality of the internal channel surface but also an increase in the acceptance level, since spoilage through end-cropping is reduced.

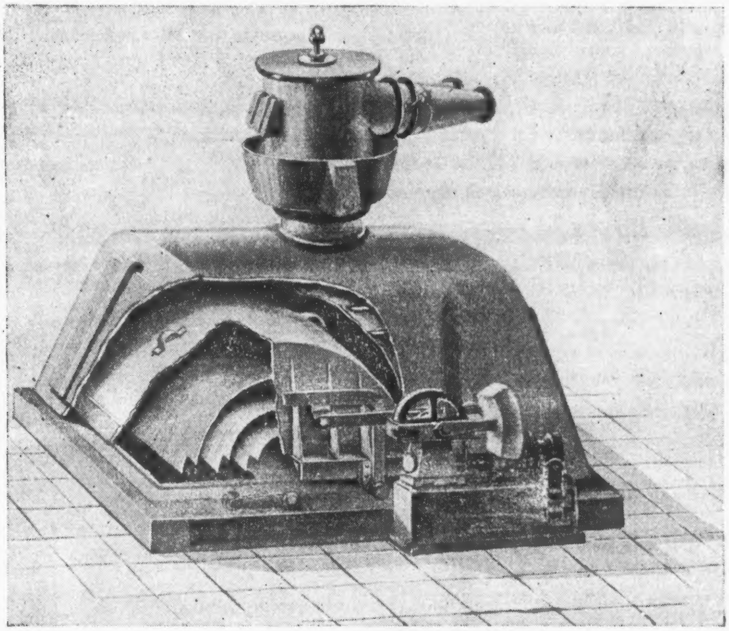
NEW TECHNIQUE AND RATIONALIZATION

NEW BUTTERFLY VALVE DESIGN

I. B. Goldenberg and E. I. Dikshtein

Magnitogorsk Metallurgical Combine

The old design of butterfly valve takes the form of an ellipsoidal chest in which the products of combustion change their direction to 180° and enter the channel leading into the common flue. It was established by investigations of the gas movement, carried out in the laboratory on valve models, that the cause of the high resistance of the chest is its unsatisfactory shape and the lack of means for lowering the resistance to the gas flow in returning. The cross section of the chest (on the majority of 190-ton Magnitogorsk furnaces) is equal to 1.37 m^2 and the cross section of the exhaust vent 1.54 m^2. Consequently, in addition to turning, the flow of the products of combustion in the chest is also constricted.



Butterfly valve with valve chest having guide vanes.

It is known that the resistance to gas flow in bends can be considerably reduced by employing guide shields or blades. Until recently, however, guide shields have not been employed in steel plant technology on the assumption, apparently, that their strength would be very small at 600-800°.

Tests on guide shields 14 mm thick made from grade ST.3 steel fitted in the chest of a butterfly valve at one of the open-hearth furnaces in the steel works have shown that fears regarding warping, jamming with dust and so on, are unfounded. The high stability of the chests with guide vanes are the main reason for their wide application at the Magnitogorsk open-hearth furnaces, the majority of which employ butterfly valves in the gas circuit.

Tests on models for studying the characteristics and design features of the butterfly valve have shown that the smallest resistance is provided by a valve with three guide vanes. Tests have also confirmed that the guide vanes should not be arranged at equal distances from each other.

As shown in the diagram, the valve chest takes the form of a welded construction with stiffening ribs to increase the mechanical strength. On the outer surface of the valve chest there are handles for ease of transport and setting up. At the lower part of the valve chest the guide vanes are inter-connected by means of sheet metal.

Repeated measurements carried out at furnaces in operation have established that in a valve chest with guide vanes, the differences in the vacuum in returning was reduced from 5-8 to 1-2 mm H_2O.

It has been established on models and on actual furnaces that the valve chest with guide vanes allows a somewhat greater quantity of products of combustion along the gas line, and that the surface temperature of the gas regenerator checkers is raised. If, before fitting the new valve chests, the surface of the gas checkers was 1180-1260°, on average, after the campaign, on the same furnaces after fitting the new design valve chest, the surface temperature of the gas checkers rose to 1240-1300° without increasing the cross-sectional area at the entry to the gas intake.

On raising the heating temperature of the checkers the degree of heating of the mixed gas is raised. The soot content of the gas is raised. The luminosity of the flame is appreciably improved. It was therefore possible to reduce the quality of gas fed into the furnace and also to limit the consumption of carburetor tar. After installing the new design valve chest the coke-gas consumption was reduced on average by 2.8% per campaign. In this way reduction in gas and tar consumption was achieved without reducing the productivity of the furnace.

At the Magnitogorsk open-hearth furnaces there are in operation ten valve chests with guide vanes, the stability of which is very satisfactory at the present time. Regular inspection of the valve chests has shown that warping of the guide vanes does not occur and that the negligible quantity of dust precipitated on the inner surface of the vanes can be readily removed during cold repair.

The installation of valve chests with guide vanes in the gas valve improves the thermal operation of the furnaces. This type of valve chest can be recommended for all open-hearth furnaces having butterfly type valves.

POURING CRANE LOAD LIMITERS

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Kharkov Tractor Plant

A reliable over-load protection for cranes considerably increases the working safety of the operating personnel and increases the life of the separate components, units, and the crane as a whole.

The work of electric gantry cranes is extremely varied. In association with this, a lack of knowledge of the weight of the load to be lifted is, in a number of cases, the cause of frequent considerable over-loads on the cranes. Pouring cranes, of course, do not have load limiters. This results in frequent considerable over-loads, carrying with them failures, accidents and prolonged crane stoppages.

Since 1954 the pouring cranes on the foundry plants at the Serge Ordzhonikidze Kharkov tractor plant have been fitted with eccentric load limiters which quickly gained wide acceptance. Operation of cranes fitted with the limiter showed that the limiter must be fitted with an indicator showing the weight of the load lifted. For example, during pouring the liquid metal, the ladle is supported at full weight by the crane; in order, under these conditions, to avoid over-loading of the crane it is necessary to determine the maximum permissible weight of the ladle by means of the auxiliary signalling device.

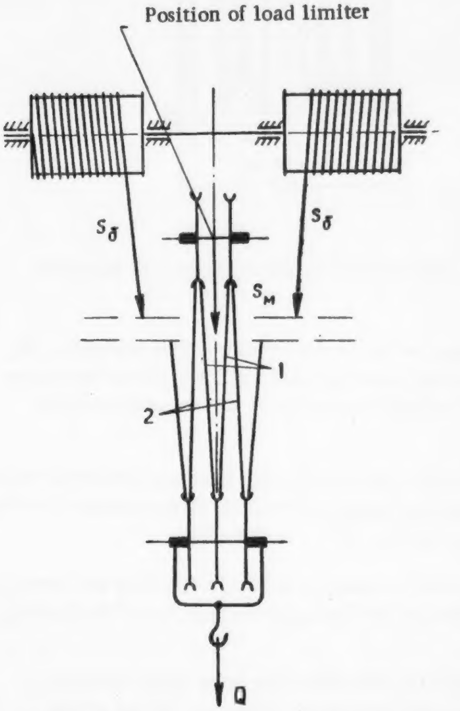

Fig. 1. Hook suspension arrangement and limiter equipment:
1 and 2) hoist cables.

The buzzer or lamp signal tells the crane driver and the operating personnel that the crane is lifting a load (the hook is taking a force), the value of which is equal to the lifting capacity of the crane, i.e., Q. The load limiter is usually adjusted for force slightly in excess of the rated lifting capacity of the crane, or equal to 1.1 Q. As the load increases above this limit the limiter is activated.

On a 15-ton pouring crane working on DC at the Kharkov Tractor Plant (KhTZ) foundry an eccentric load limiter was fitted, the principle of which comprises a universal mechanism (for constant lifting-capacity cranes) and which can be employed independently of the crane-block hook-suspension system.

The limiter is fitted on the crane at the hoisting cable across the two fixed blocks (Fig. 1), which carry the first and second hoist cables. Both blocks are located on the eccentric roller of the limiter and transmit part of the force from the weight of the hoist onto the shaft across the arms of the hoisting cable.

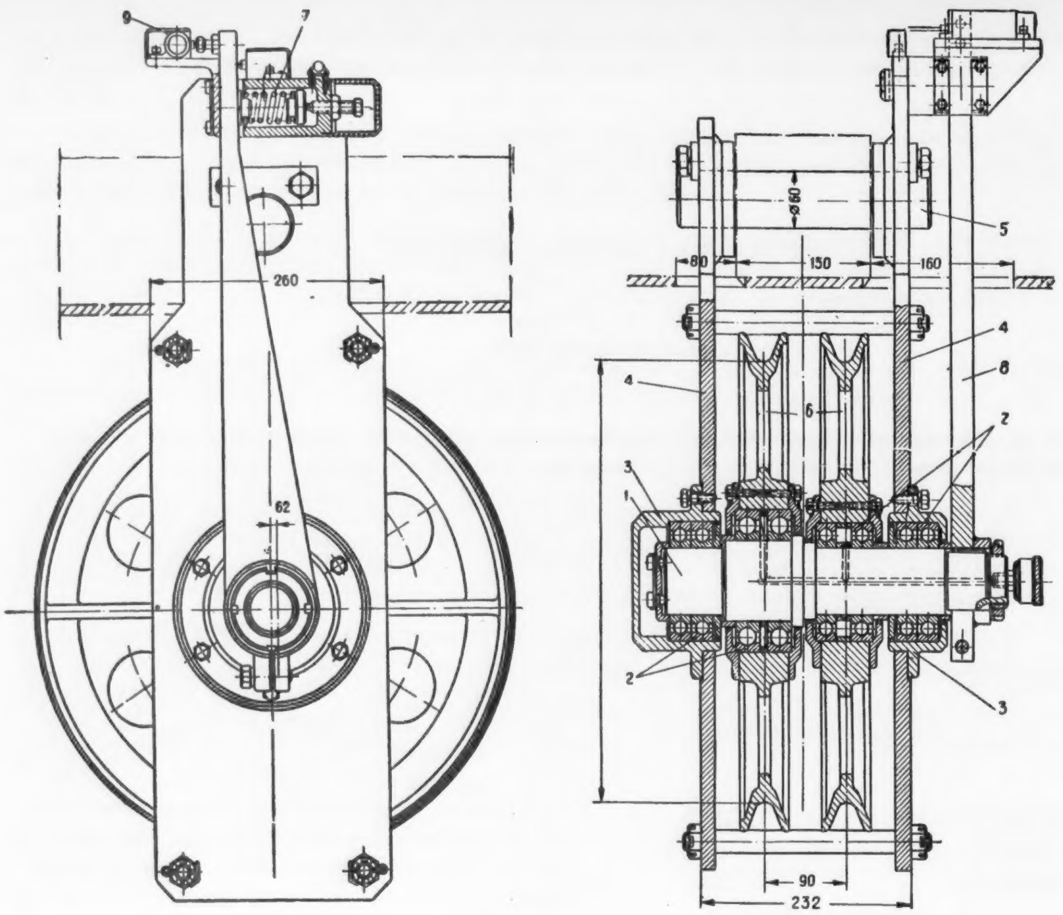

Fig. 2. Load limiter for electric gantry crane Q = 15 t:
1) eccentric shaft; 2) ball-bearing; 3) hooks; 4) jaws; 5) axis of the hoist winch frame; 6) freely rotating blocks; 7) springs; 8) lever; 9) final switch.

The eccentric shaft (Fig. 2) is supported on four ball-bearings, set up in the housings. The two hooks are fixed in the two jaws of the limiter which are joined at a given distance by cotter pins and nuts. The jaws of the limiter are located on an axle mounted in the frame of the hoisting winch. Two blocks are set up on the eccentric shaft in ball-bearings rotating independently of the shaft rotation.

At the projecting end of the shaft a lever is mounted vertically, the other end of which is pressed against the support by a spring. The spring tension is regulated by means of a bolt. The projecting part of the lever may open and close the contacts of the controlling circuit breakers (namely, the hoist limiter and the alarm) as it rocks.

The eccentric jaw of the shaft on which the block is mounted is arranged in such a way that the direction of eccentricity is taken up by the perpendicular resultant of the sum of the forces in the branches of the hoisting cable (Fig. 1).

Hence, on filling the ladle with metal up to a given weight the counterweight lever of the limiter, compressing the spring, begins to rotate relative to the axis of the eccentric shaft. When the weight of the ladle is equal to the hoisting capacity of the crane, the limiter lever rotates through the necessary angle and frees the depressed handle of the alarm switch. At the same time the contacts are brought together and the alarm circuit is closed, thus bringing into circuit two (for safety) parallel signalling lamps in the crane driver's cabin.

On further increasing the weight of the ladle the angle of rotation of the lever also increases. When the load exceeds the hoisting capacity of the crane by 10%, the lever depresses the hoisting switch and closes the contacts, bringing in an intermediate relay which, on activating opens hoist contacter circuit. This stops the hoist motor. Further lifting is impossible until the weight of the load on the crane hook is reduced to the hoisting capacity. The hoist limiter with indicator alarm has operated already for two years without failure. It is accurate in operation, simple in construction, straightforward in preparation and control. The cost is relatively small.

For operating cranes on AC the hoist limiter is connected into the hoist control circuit, i.e., in series, with the overwind cut out.

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SCHOOLS OF ADVANCED EXPERIENCE

MORE ATTENTION TO THE TRAINING OF THE WORKERS IN THE LEADING CRAFTS

L. A. Rimsky
U. S. S. R. Iron and Steel Ministry, Labor, Training and Wages Department

Considerable work has been carried out during the last three years in the iron and steel industry for improving the training and raising the skill of the workers in the leading crafts.

During 1953-1954 training was carried out at the Magnitogorsk Metallurgical Combine on almost the entire labor force of furnace men and assistant furnace men, gas fitters, steel smelters and assistant steel smelters, foundry men, rollers and welders.

In 1953-1955 at the "Azovstal" plant 90% of the blast-furnace men were trained: 79% of the open-hearth workers and 78% of the rail-mill workers.

In 1956 tests were carried out by the Institutes of Technical Training on the state of training work among the leading crafts. The tests showed that the training position at a number of factories was by no means satisfactory. Take, for example, the way in which technical training of steel smelters is undertaken. The steel smelter must be familiar with the technological process of manufacture of various grades of steel, the design and correct operation of the furnace and also the measurement and control equipment.

Modern technique in the field of steel smelting (the use of oxygen, the changeover of furnaces to increased charge, automatic control of steel smelting production and so on) requires a continuously rising level of skill from the steel smelter.

Sample studies of the steel smelter crews on open-hearth furnaces together with the duration of working shows that at individual concerns the steel smelters have neither sufficient experience of the work nor the necessary knowledge.

factory	total smelters	experience of smelters							
		up to 1 year		1-4 years		5-10 years		over 10 years	
		number of workers	% of total	number of workers	% of total	number of workers	% of total	number of workers	% of total
Magnitogorsk steel works	88	1	1.1	12	13.6	16	18.3	59	67.0
Azovstal	42	–	—	10	23.8	22	52.4	10	23.8
Zaporozhstal	38	2	5.3	15	39.4	21	55.3	—	—
Makeyevka Kirov	73	21	28.8	25	34.2	14	19.2	13	17.8
Voroshilov	28	3	10.8	21	75.0	4	14.2	—	—
Dzerzhinsky	51	6	11.8	37	72.5	5	9.8	3	5.9

The length of experience of the steel smelters on the open-hearth furnaces of individual concerns is set out in the table.

The table shows that at the Magnitogorsk steel works and at the Azovstal and Zaporozhstal plants, a clear majority of the smelters have experience in the trade of five years and over.

At the Voroshilov plant only 14.2% of the smelters have a similar degree of experience, at the Dzerzhinsky plant 15.7% and at the Kirov plant 37.0%; the majority of smelters have experience at the trade of not more than four years. It is not surprising that smelters with small experience frequently upset the established technology (thus injuriously affecting utilization of the units) increase the stoppages and allow spoilage.

There are several excellent masters of the craft among the smelters, workers in the van of production such as Rodichev (Magnitogorsk steel works), Vidaschuk (Kuznetsk steel works), Martynov (Zaporozhstal), Maletsky (Azovstal), Lukyanov (Novo-Tagil plant) and others. It is essential that the experience of the best should be available to all the smelters. In this way it would be possible to substantially raise the productivity of the smelting plants.

At the Kuznetsk steel works close attention is turned to the study and passing on of advanced experience. A complex plan of study is worked out daily with the dissemination of advanced experience, indicating the sections, trades and names of the leading workers whose experience should be propagated, the names of works studied, dates and responsible executives. Such a plan facilitates the best form of study and the introduction of advanced experience.

Inter-plant schools are set up at the works for studying and disseminating advanced methods of work.

By way of example there has grown up a tradition of exchange of experience through the school between No. 1 and No. 2 open-hearth furnaces. In this way a study of the advanced experience of assistant smelters Kuznetsov and Melnikov made possible a reduction in the stoppage for repairing the tap holes in both open-hearth plants.

However, the most important feature in training is that the workers acquire deep and intimate knowledge and systematically increase their skill. Therefore, alongside the schools of advanced experience there must be a wide expansion in technical production courses in schools for the masters of socialist labor. These possibilities are unfortunately not exploited at all the factories.

At the Voroshilov plant in 1954-1956 the workers of the leading crafts at the open-hearth plant (smelters and assistant smelters, chargers, crane drivers, tappers, mixers and ladle men) carried out training not in technical production courses but only in general short courses and in schools of advanced methods of labor.

At the Chelyabinsk steel works during the last three years of the steel smelters and assistant smelters, only the following took technical production courses: No. 1 open-hearth plant (42 out of 90), at the No. 2 open-hearth plant (49 out of 94), at the electric steel smelting plant (48 out of 83). During the last three years the bricklayers on lining ladles and troughs did not take any training.

Several incorrectly assume that once the workers have completed the technical school and the factory training school, no further training is necessary. The technical production courses are the follow-up for raising the skill of workers having passed individual team training for production or having completed the factory training school and the technical school.

One serious shortcoming standing in the way of higher skill on the part of the workers in the leading trades is the assumption by these workers of higher positions without a knowledge test and without taking into account the results of technical training.

In order to improve technical production training of the workers in the leading trades it is necessary, first of all, to organize training of workers unfamiliar with production technology, the principles of operation and maintenance of equipment. It is most advantageous for workers of the leading trades to undertake training on extended technical production courses or in $2^1/_2$ year artisan schools.

The technical and engineering workers of the plants must, in the process of the day to day work, train the workers correctly in carrying out all the technological operations, assist them to gain knowledge and introduce advanced methods of work.

The training of the workers is one of the decisive factors in increasing the productivity of labor and achieving success in the struggle for fulfilling the Five-Year Plan ahead of time.

It is necessary to strive consistently for each worker of the leading trades to raise the level of technical knowledge and skill. In this way it will be possible to bring out and more fully utilize the reserves for raising the productivity of labor which exist in every plant.

STATE AND PERSPECTIVES OF THE MANUFACTURE OF PROCESS OXYGEN

Doctor of technical sciences, I. P. Usyukin

Differentiation is made in industry between commercial gaseous oxygen used for arc welding and technological gaseous oxygen used for intensifying industrial processes (chemistry, metallurgy and so on).

Germany was the foremost country in the manufacture of technological oxygen during the Second World War. The construction of plants for the production of process oxygen was begun here in the thirties by the firm of Linde from the moment of Frenkel's invention, which proposed instead of the usual continuous tubular heat exchange equipment (recuperators) periodic equipment consisting of an aluminum charge (regenerator). Up to 1945, 80 process oxygen plants had been built from the Linde-Frenkel patent with an overall production of 160,000 m^3/h, the production of an individual plant being 1700-3600 m^3/h. During the post-war period in western Germany this firm tuned up the output of technological oxygen plants,and at present the over-all production of the German plants is 350-38,000 m^3/h. A considerable part of the oxygen is utilized in the chemical industry and coal-oil industry together with a large fraction in metallurgy.

The U. S. A. started building process oxygen plants only in 1945. In the main, use was made of the German experience and such firms as Linde Air Products, together with some others, set up plants on the Linde-Frenkel system with an output not greater than 5000 m^3/h. Hydrocarbon Research Inc. (Staycey Dresser) built four plants with an output of 29000 m^3/h, built in pairs at two factories. After starting up, however, they were stopped and are not, at present, working. The Blaw-Knox firm has turned out a plant with an output of 4500 m^3/h, differing from the Linde system in that argon is produced at the same time as oxygen. The total output capacity of the technological oxygen plants in the U.S.A. in 1952 was estimated at 220,000 m^3/h including four plants, not working, of 29,000 m^3/h.

The French firm Aire Liquide has in project a technological oxygen plant carrying the name Oxytone. Technological operating data does not exist for these plants. Recently Great Britain has also commenced the construction of a process oxygen plant.

The Soviet Union was close on the heels of Germany in 1935 in starting the construction of process oxygen plants. In 1939 a plant was started at the Dnepropetrovsk steel works with an output of 5000 m^3/h of 60% oxygen, but this was destroyed during the war.

In the main, the construction of process oxygen plants was commenced in 1946. Up to the present time several types of plant have been set up, including the 300-2D with a production capacity of 300 m^3/h of 99% oxygen ; KT-1000, with a capacity of 1000 m^3/h of 99% oxygen ; BR-4, BR-3 with capacities of 3500-5200 m^3/h of 99% oxygen, and the BR-1 with a capacity of 10-15000 m^3/h of 96-99% oxygen.

The BR-1 and BR-3 plants were designed and constructed in 1947-50 only for low-pressure operation and without chemical purification of the air for carbon dioxide and moisture. They employed the original principle of the triple blast in the regenerators as developed by Soviet experts. Both these plants have been operating for a long time and are very efficient.

By the end of the fifth Five-Year Plan, process oxygen plants were built in the U.S.S.R. with a total capacity of approximately 70,000 m^3/h. The sixth Five-Year Plan provides for the construction of twenty new oxygen plants and a considerable extension of the existing plants. By 1960 the capacity of the plants will reach 460,000 m^3/h, that is to say almost $1\,^1/_2$ times the capacity of all the process oxygen plants constructed in Germany during the last twenty-six years.

In recent years a number of mistakes have been made in the construction of oxygen plants by Soviet metallurgists, gas engineers and oxygen engineers. Thus, for example, the largest oxygen plant of the Novo-Tulka metallurgical plant is exploited over a long period at 30-40%, while the remaining plant is idle; the second plant in order of capacity at the Shchekinsk chemical gas works is also exploited to approximately 40% while the remaining plant is idle and is falling out of use. Hence, the existing oxygen plant capacity is exploited only to a limited extent. On the other hand, the Azovstal and Zaporozhstal oxygen plants are not big enough and must be extended.

In building new oxygen plants at the Glavkislorodmash and the Giprokislorod, the Ministries of Iron and Steel and Nonferrous Metals must take into account the error tolerances, otherwise the solution of the oxygen blast problem in metallurgy loses all sense and the economic effect of employing oxygen becomes a negative one.

KT-3600 plant. With this plant part of the air is compressed to 120-160 atm. abs. This requires massive and complex piston compressors, pipe-lines and auxiliaries operating at high pressure. The air has to be chemically cleaned for carbon dioxide and the moisture removed with special ammonia equipment. The capital expenditure on this type of plant is very high. The amount of shift labor and the energy consumption for maintaining the large amount of complex machinery and equipment is much greater than at other plants. The large KT-3600 plant cannot therefore, continue to be put to use.

The Blaw-Knox has some advantages over others. Instead of employing aluminum tape the regenerators are filled with a special brick checker, capable of operating at a low temperature. In order to reduce the cost of the oxygen, provision is made for obtaining raw argon.

This plant, however, has not so far had a long operating run and cannot, therefore, be recommended for high-capacity plants, since it consists mainly of units with a capacity not exceeding 5000 m^3/h of oxygen.

The Linde low-pressure plant is more advanced than the KT-3600 plant since this provides for operation only at low pressure. In many respects it reproduces the constructional elements of the KT-3600 plant, as a result of which the rated capacity does not exceed 7000 m^3/h. With the old design of units the oxygen extraction coefficient from air was low, as a result of which the energy consumption at the Linde Plant is 1.2 times that for the best Soviet plants. On account of the low capacity and high energy consumption, the cost of oxygen from these plants will be higher than at the BR-1 plants. The gas absorbers for removing carbon dioxide and acetylene from the air are not reliable in operation and do not allow long-term uninterrupted operation of the plant.

The BR-1 plant (Fig. 1) is the largest of the plants operating and consists entirely of new highly-efficient units providing complete uninterrupted removal of carbon dioxide by utilizing the internal thermal capacity of the system. The plant is fully protected against the entry of acetylene into the separator section, and safe operation is guaranteed. The rectifying plates of the separating apparatus assure maximum oxygen extraction coefficient.

By employing the internal thermal capacity of the cycle for carbon dioxide removal without external interference, a long-term continuous operating period is provided which is adequate for the needs of the user.

The method of operating this plant is as follows.

After removing mechanical additions through a filter, the air is compressed with a turbo-compressor to 5.8 atm. abs. The compressed air is distributed between nitrogen and oxygen regenerators, where it is cooled to the saturation state. For removing water and carbon dioxide the air is introduced to the oxygen regenerators 3% less by weight then the obtained production oxygen. The remaining part of the air is passed to the nitrogen regenerators.

The heat exchange in the nitrogen regenerators follows the following sequence: the reverse nitrogen stream provided in the separating apparatus passes through the super-cooler and arrives at the automatic nitrogen valve along the pipeline, continuing to the regenerator checker, bringing it down to a low temperature. Part of the air is taken from the cooled air collector and arrives immediately after the nitrogen through the booster valve in the low part of the regenerator in the form of a reverse flow, passing from the cold to the hot end of the regenerator, taking up heat from the checker. The reverse-flow air enters through the booster valve and the gas-engine heat exchanger into the lower column of the separating apparatus. The once-through air is then passed through the regenerator, being cooled from the nitrogen and the reverse-flow of the gas-engine air.

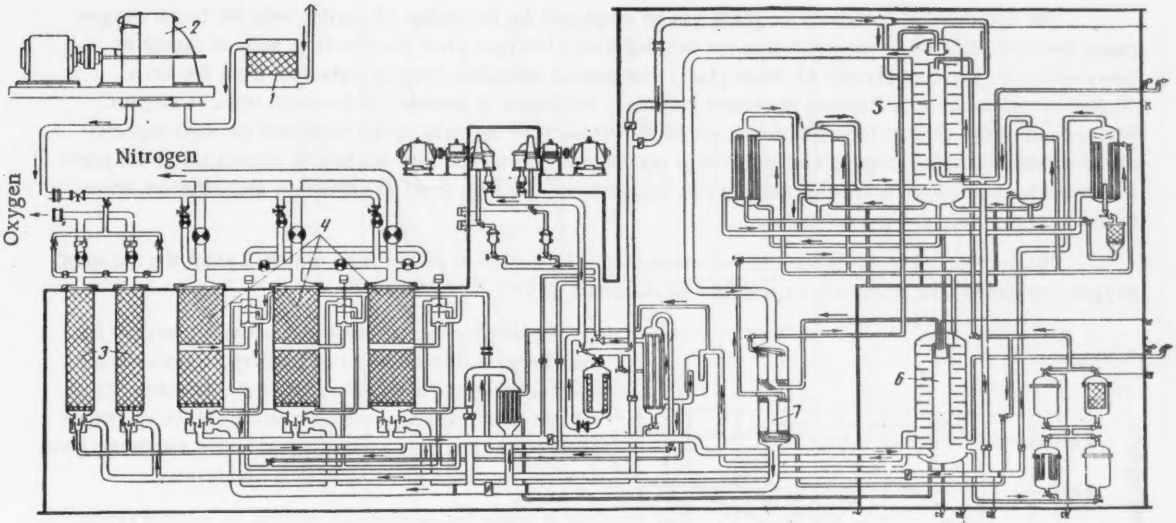

Fig. 1. Plant for the production of process oxygen type BR-1:
1) air filter; 2) turbo-compressor; 3) oxygen regenerator; 4) nitrogen regenerators; 5) upper rectifying column; 6) lower rectifying column; 7) liquid nitrogen and air super-cooler.

Figs. 2, 3 and 4 illustrate the process plant operating system, constructed on the basis of long-term tests at the plant.

Figure 2 shows that the productivity of the plant falls within the wide range of 7000 to 15200 m^3/h with a high oxygen concentration (over 96%) and nitrogen concentration (99.2%).

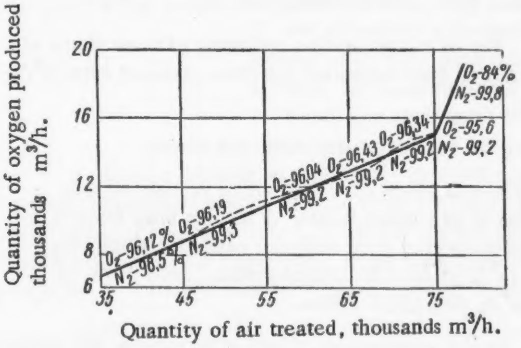

Fig. 2. Relationship of plant capacity to quantity of air treated:
1) based on 1953 experience; 2) based on 1954 experience.

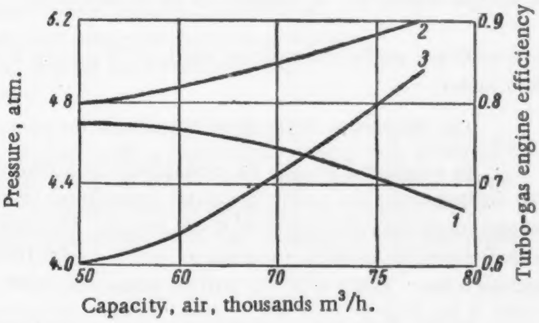

Fig. 3. Relationship of gas-engine turbo efficiency to quantity of air treated:
1) turbo-gas engine efficiency; 2) pressure before turbo-gas engine at the throttle; 3) pressure before the turbo-gas engine after throttling.

In view of the thermal deficiency with this plant the air pressure has to be reduced by throttling from 0.5 to 0.8 atm. before the turbo-gas engine, thus indicating the possibility of further obtaining raw argon and krypton from these plants. On lowering the capacity of the plant and the purity of the oxygen the energy consumption falls off sharply, reaching much smaller values than the previously mentioned foreign plants (Fig. 4).

The constructional units of the plant can be employed for the design of similar units for larger oxygen plants thus providing some perspective in the development of oxygen plant construction, such as cannot be provided by the previous plants. At these plants with almost complete oxygen extraction from the air, a krypton-xenon mixture is obtained at present and argon extraction is provided at the same time. Complex break-down of the air into its components, the high efficiency of the units of the plant and the high capacity afford minimum capital expenditure and energy consumption. With the high productive capacities of the plant it is possible to employ the most powerful turbo-compressors with high level of efficiency and efficient drives in the form of steam and gas turbines.

The over-all advantages enumerated mean that it is possible to obtain from the BR-1 plant the cheapest oxygen, combined with minimum expenditure of materials in their erection.

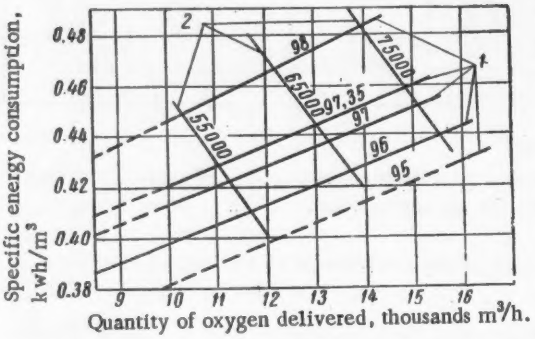

Fig. 4. Relationship of specific energy consumption to quantity of oxygen delivered and purity:
1) oxygen purity, %; 2) quantity of air treated m^3/h.

In tackling the problem of introducing oxygen into metallurgy and other fields of industry, account has not been taken of the economic side of the question. This has led to the position where incorrect choice of oxygen plant equipment may prove economically disadvantageous in some processes where oxygen is employed.

A graph has been drawn up (Fig. 5) demonstrating the correct choice of oxygen plant, providing minimum capital expenditure, minimum oxygen cost and minimum building volume and area, on the basis of project data.

The capacity of plants in the oxygen industry proceeds in multiples of three; plants are made for 6-7, 30, 100, 300, 1000, 3600 and 12-18000 m^3/h. With a triple increase in the capacity of plants up to 1000 m^3/h, there is a saving in capital expenditure in the ratio of 1.4: 1. With a triple increase in the capacity of plants above 1000 m^3/h the saving amounts to 1.7-1.8: 1.

For an oxygen station consisting of three plants of 1000 m^3/h the capital expenditure per 1 m^3 of oxygen is equal to 4500 rubles, and for three plants of 3600 m^3/h-2400 rubles.

The diagram is of practical improtance for selecting the type of oxygen plant and station.

By employing oxygen, the productivity of a blast furnace is raised by 25%. With a furnace costing 150 million rubles, the saving in capital expenditure amounts to 38 million rubles. A modern blast furnace rerequires more than 25-30,000 m^3/h of oxygen. The cost of an oxygen plant with this capacity, employing various plants for producing gaseous oxygen is: with 1000 m^3/h plants, 145 million rubles; 3600 m^3/h, 75 million rubles; 11000 m^3/h, 42 million rubles and 33000 m^3/h, 24 million rubles.

Hence, from the standpoint of capital expenditure, for the manufacture of iron using oxygen, the oxygen plants serving the blast furnaces must be built in units of 10-30000 m^3/h.

There is a similar relationship for the cost of oxygen. Raising the capacity of small units (up to 1000 m^3/h) in the ratio of 3: 1 lowers the cost of the oxygen in the ratio of 1.2: 1, while raising the capacity of large units (above 1000 m^3/h) in the ratio of 3: 1 reduces the cost of the oxygen in the ratio of 1.5: 1.

It can be seen from the graph that the cost of oxygen with electricity costing 14 kopeks/kwh is 13 kopeks/m^3, with electricity costing 10 kopeks/kwh, 13 kopeks/m^3 and 5 kopeks/kwh, 10 kopeks/m^3.

The cost of oxygen is higher from the project data, since the projects did not envisage full automation of the oxygen manufacturing process, the reduction in labor costs and combined turbo-compressors with steam drive.

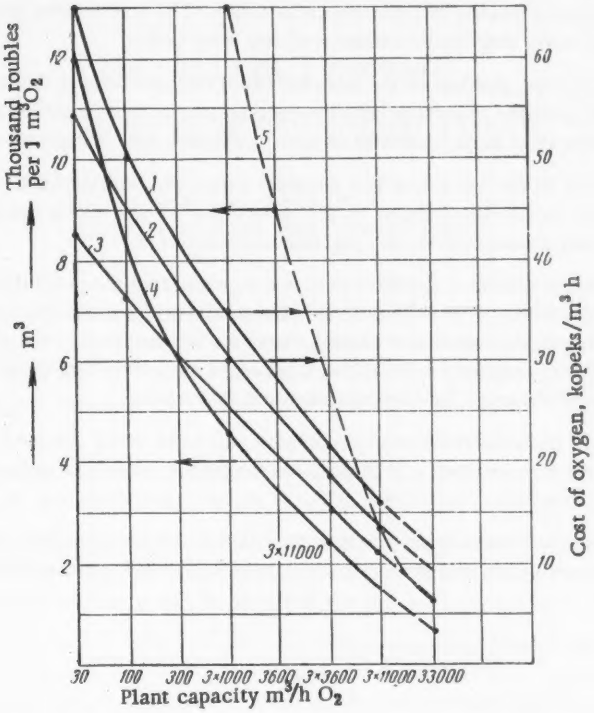

Fig. 5. Relationship of plant capital outlay and cost of oxygen to plant capacity:
1) cost of oxygen with electrical energy costing 14 kopeks/kwh, kopeks/$m^3 O_2$; 2) ditto, electrical energy costing 10 kopeks/kwh;
3) ditto, for 5 kopeks/kwh; 4) capital outlay per m^3/h O_2;
5) building volume per 1 m^3/h O_2, m^3.

In the case of large automatically operated oxygen plants with a productivity greater than 10000 m^3/h, the cost of electrical energy will comprise 65-70% of the total cost of the oxygen; with an electrical energy consumption of approximately 0.35 kwh/m^3 O_2 and cost of electrical energy 5 kopeks/kwh, the cost of the oxygen will fall to 2.5 kopeks/m^3. In the case of multiple break-down of the air, yielding krypton and argon, part of the cost will be carried by these other products.

On introducing oxygen into the blast furnace process the calorific value of the blast-furnace gas is considerably raised, approaching the calorific value of the mixed gas employed at the present time for open-hearth furnaces. It is therefore no longer necessary to employ coke-oven gas for mixing with the blast furnace gas. The freeing of the coke-oven gas at the steel works and the availability of by-product nitrogen from the oxygen plants open up the possibility of very cheap synthetic ammonia production, oxygen then appearing as a by-product.

The data set out in the graph would require some refinement but the principles demonstrated remain in force.

The following conclusions can be drawn:

1. The projected construction of a large quantity of oxygen plants and installations during the present Five-Year Plan must be directed toward the introduction of the most advanced technical solutions. There is no justification for the construction of several score of the costly KT-3600 plants. The construction of these plants would result in the steel works being fitted with out-of-date equipment of a type which has been abandoned by foreign manufacturers; their operation and the high cost of oxygen would render the application of oxygen for the intensification of a number of processes unfavorable. The construction of oxygen plants in this manner would turn out more costly than the construction of new steel works.

2. A radical solution of the problem of the introduction of oxygen-blowing into the steel manufacturing processes can be applied with positive advantage only by erecting large oxygen installations with a capacity of 1200-30000 m^3/h and for large steel works—a central oxygen installation with a capacity of 50000 m^3/h.

Plants have been set up in the U.S.S.R. with a capacity exceeding foreign plants by 3-4 times and providing cheap oxygen with low capital expenditures on the installation. These plants render it possible to supply oxygen to all the metallurgical processes, including pig-iron manufacture.

3. Operating experience with BR-1 and BR-3 units has demonstrated the possibility of constructing even larger units with a capacity of 30000-50000 m^3/h. These plants provide the possibility of employing a low air pressure, of reducing the energy consumption and capital outlay for the installation and of providing the cheapest oxygen. The construction of large capacity units allows large axial and centrifugal turbo-compressors to be used in conjunction with economical drives in the form of steam and gas turbines.

4. The cost of oxygen is considerably reduced, krypton and argon being obtained at the same time. Hence, the separation units must be designed and constructed for the combined extraction of oxygen, argon, krypton and other gases from the air.

The installation of large process oxygen plants in the U.S.S.R. opens up the way for the widespread introduction of oxygen into the various industrial processes and particularly into the iron and steel industry.

METALLURGY ABROAD

BLAST-FURNACE PRODUCTION ABROAD

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Manager Gipromez Blast Furnace Section

General characteristics. In the period 1945-1955 the manufacture of pig-iron rose in all the main capitalist countries (Fig. 1). The first place for pig-iron manufacture is occupied by the U.S.A. Mean rates of increase in the manufacture of pig-iron are as follows: U.S.A., 2.15 million tons per year; Federal German Republic, 1.58 million tons per year; France, 0.98 million tons per year; Great Britain, 0.53 million tons per year.

The variation in the number of blast furnaces is shown in Fig. 2. The increases in the total volume of blast furnaces in the U.S.A. and the Federal German Republic are set out in Table 1.

The increase in pig-iron manufacture in Great Britain and the Federal German Republic was accompanied by a reduction in the number of blast furnaces. The same occurred in the U.S.A. in the early post-war years, but the number of blast furnaces subsequently increased in the U.S.A.

TABLE 1

Increase in the Total Volume of Blast Furnaces, Thousands m^3

Country	Year			
	1945	1946	1951	1954
U.S.A.	204.5*	—	218.7*	249.4**
FGR	—	appr. 81	—	—

* the useful volume is determined from N. A. Pavlov's formula:

$$V = 0.54\ DH.$$

** the useful volume is determined from the formula:

$$V = 1.1\ V_e$$

where V_e is the effective volume of the furnace.

The maximum blast-furnace volume rose to 1810 m^3 in the U.S.A., 1745 m^3 in Great Britain and 1500 m^3 in the Federal German Republic.

The average blast-furnace volume rose through the construction of large blast furnaces and the reconstruction of existing furnaces (Fig. 3).

The increase in blast-furnace volume in the U.S.A. through reconstruction in the period 1945-1953 was:

Number of reconstructed furnaces 43*
Increase in volume m^3. 12112

Data showing the consumption of the basic raw materials per 1 ton of pig-iron are set out in Table 2.

The agglomerate fraction in the blast-furnace charge was increased (Fig. 4).

Attention should be turned to the substantial use of pyrites ash in the agglomerate charge in the Federal German Republic (2.7 million tons for 9.2 million tons of agglomerate in 1952). In 1953 approximately 12% of all the pig-iron was manufactured in the Federal German Republic from pyrites ash. In the U.S.A. approximately 0.6 million tons of pyrites ash was employed.

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[*] Disregarding major overhauls with a small increase in the volume.

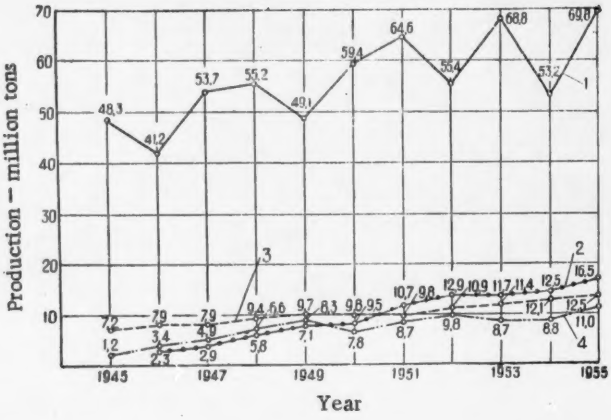

Fig. 1. Increase in pig-iron production in the principle countries: 1) U.S.A.; 2) Federal German Republic; 3) Great Britain; 4) France.

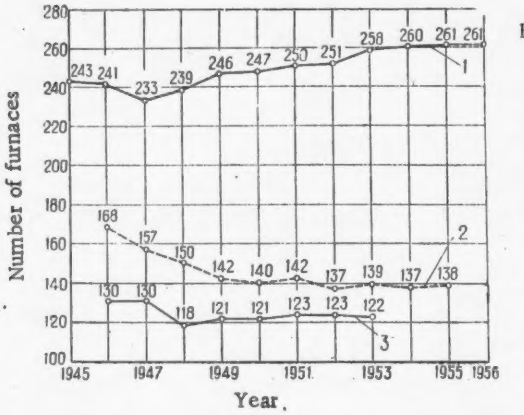

Fig. 2. Variation in the number of blast-furnaces 1) U.S.A.; 2) Great Britain; 3) Federal German Republic.

In Sweden, Federal German Republic and Great Britain fluxed agglomerate is employed.

In the Federal German Republic and Great Britain agglomerate is produced from domestic fluxed ores. Experiments are being carried out in the Federal German Republic for producing fluxed agglomerate with a limestone addition initially baked on the sintering machine to a 6-8% carbon dioxide content. The use of a fluxed agglomerate with a basicity of 0.72 provided a reduction in the coke consumption on average of 65 kg/t iron; it was possible to lower the basicity of the slag from 1.35 to 1.25-1.20 without raising the sulfur content of the iron.

Blast-furnace operation with high gas pressure in the throat has become common. Fourteen furnaces are operating on this system in the U.S.A. and three in Great Britain.

The gas pressure in the throat on American blast furnaces does not exceed 0.9 atm. and in Great Britain 0.4 atm.

For the purpose of intensifying the smelting process, oxygen-enriched blast is being introduced in foreign blast-furnace practice.

In the post-war period a low-shaft furnace producing ferro-manganese has been operating with an enriched-oxygen blast at the Gutehoffnungshutte plant in Western Germany. Oxygen was first introduced into the hot blast and subsequently into the cold blast. The blast temperature was at first 650° and subsequently 850°. The furnace operated smoothly. The following are theoperating characteristics of the furnace:

Oxygen content in the blast, %	21.0	31.5
Productive capacity:		
t/day	35.2	48.1
%	100	129
Dry-coke consumption, kg/t pig-iron	2036	1579
Intensity of driving, kg/m^2. h	733	779
Temperature, °C:		
blast	854	641
throat	459	346
Pig-iron composition, %:		
silicon	1.0	0.5
manganese	43.9	44.2

In the U.S.A. the use of oxygen in blast furnaces was commenced immediately after the war for smelting ferro-manganese. The advantageous results were confirmed with the further application of a higher oxygen concentration in the blast.

The following results were obtained:

Oxygen content in the blast, %	23	25	27	29
Duration of operation, days	10	66	22	7
Increase in productivity, %	15	27	38	46
Reduction in coke consumption, %	3	7	12	8

At the Werton Steel plant an enriched oxygen blast has been employed since February, 1951 for smelting conversion pig-iron.

The oxygen is distributed uniformly between two furnaces with volume 1200 and 1262 m^3 and two furnaces with a volume of 1427 m^3.

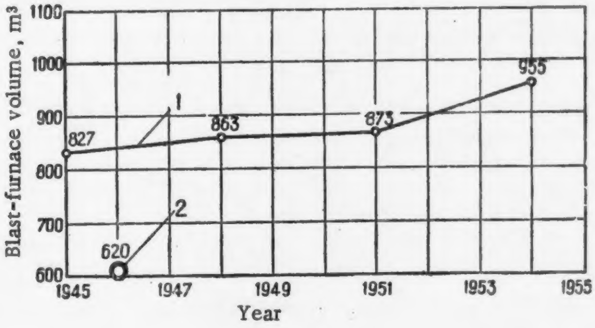

Fig. 3. Variation in the mean blast-furnace volume;
1) U.S.A.; 2) Federal German Republic.

The humidity of the blast is regulated in accordance with the degree of oxygen enrichment.

Oxygen enrichment of the blast, %
1.0 1.5 2.0 2.5 3.0

Humidity of the blast, g/m^3
11.5 13.8 16.0 18.4 20.7 to 23.0

The steam is considered as a supplementary cheap source of oxygen. The mean average operating characteristics of No. 2 blast-furnace at the Werton Steel plant for one month's operation are set out below.

Degree of oxygen enrichment of the blast, %	1.5
Humidity of the blast, g/m^3	13.8
Blast consumption, m^3/min.	2120
Temperature °C:	
hot blast	645
throat	155
Quantity of agglomerate in the charge, %	22.5
Quantity of large lumps of ore in the charge, %	10
Productivity of the furnace, t/day	1294
Blast-furnace dust carry-over kg/t pig-iron	36.3
Coke consumption kg/t pig-iron	757
Composition of the pig-iron, %:	
silicon	1.1
manganese	1.95

The conclusion was reached at the Werton plant that the application of the oxygen-enriched blast raises the iron yield (4.75% for each 1% increase in oxygen content in the blast), reduces the carry-over of blast-furnace dust, reduces the throat temperature by 28% and the coke consumption by 12.5-25 kg/t of iron with a 2% oxygen enrichment of the blast.

A feature of interest is the blowing of pig-iron in the blast-furnace bath with oxygen for the purpose of refining the metal, carried out in an experimental furnace in Japan in 1954.

Only individual furnaces operate with humidified blast of constant humidity in the U.S.A.; blast conditioning (humidifying in the dry season and drying in the wet season of the year) is also practiced in the U.S.A.

Soda, either in the pure form or mixed with lime or other substances was already employed before the Second World War for the purpose of desulfurizing. However, in the case of delayed removal of the soda slag, reversal of the sulfur occurs from the slag to the metal.

Desulfurization of the iron has been carried out at the Surahammer plant (Sweden) since 1950, employing lime in a rotating furnace with a capacity of approximately 18 t and at the Fagersta plant since 1952 in a 25 t furnace. At the Surahammer plant Thomas pig is desulfurized from 0.01 to 0.007% S and at the Fagersta plant from 0.05 to less than 0.01% S.

Molten iron is poured into the furnace and 2% lime and 0.5% coke fines are added. The desulfurization process continues for 30 minutes.

Rotating furnaces are employed not only for desulfurization, but also as ladles for transporting the pig-iron.

This type of plant for desulfurizing the iron is employed also at the Domnarvet plant (Sweden).

Calcium carbide is also used for desulfurizing iron, being blown into the metal with nitrogen. The sulfur content in the iron is reduced to 0.01-0.02%. Calcium carbide removes approximately 90% of the sulfur in the metal. The calcium carbide consumption is 10-12 kg/kg of sulfur removed. The cost of desulfurizing by means of calcium carbide is higher than desulfurizing with soda.

In France, Belgium and Luxemburg desulfurization of iron has been carried out in the ladle by means of synthetic low melting-point slags. The best results were obtained with a mixture of 60% Na_2CO_3, 20% NaOH and

20% CaC_2. Using 11.5 kg/t of iron the degree of desulfurization with initial sulfur content 0.05-0.09% was 55-65%. Using only soda the degree of desulfurization did not exceed 30-40%.

TABLE 2

Consumption of Raw Materials and Fuel in the Blast Furnaces, kg/t Pig-Iron, (Including Blast-Furnace Ferrous Alloys)

Material / Year	1950	1951	1952	1953	1954
U.S.A.					
Iron-ore and agglomerate	1737	1725	1710	1691	1656
Metal additions	56	54	55	53	50
Slag, hearth cinder	110	102	110	128	153
Limestone	429	432	426	418	395
Coke	922	924	922	906	873
Agglomerate content in the charge,%	16	18	19	not given	not given
Great Britain					
Iron and manganese ore	1765	1832	1828	1807	1653
Including agglomerate	344	366	406	423	495
Metal additions	94	85	70	69	73
Slag, cinder and pyrites ash	88	86	88	98	85
Limestone	218	220	235	240	228
Coke	1028	1063	1070	1054	999
Agglomerate content in the charge,%	19.5	20.0	22.2	23.0	30.0
Federal German Republic					
Iron and manganese ore	1632	1805	1916	1914	—
Agglomerate	554	662	723	739	—
Metal additions	214	140	126	98	—
Cinder, slag	219	230	220	230	—
Limestone	179	192	251	227	—
Coke	947	983	1031	1013	—
Agglomerate content in the charge,%	83.9	36.7	37.7	38.6	—

A process has been developed in France for quick deep desulfurization of iron by blowing powdered lime into the iron by means of an inert or reducing gas through the tuyere. Tests were carried out on an experimental installation and later at the plant. Powdered lime, 0.5-0 mm was blown with nitrogen into the iron poured into a converter having a capacity of 300-2500 kg; the quantity of powdered lime in the nitrogen amounted to 10-40 kg/m^3. The tests showed that there was practically no wear of the lining and tuyeres. For purposes of simplification it was better to blow into the ladle employed for transporting the pig-iron, fitting the ladle with tuyeres. A test was carried out in a ladle with a capacity of 2.5 t. The initial sulfur content was 0.3%. Blowing lime to the amount of 2% of the weight of the pig-iron for 3-5 minutes lowered the sulfur contents in the iron to 0.01%.

The best coefficient of utilization of useful volume in the world (0.58) was achieved in Sweden with a blast furnace at the Domnarvet plant in 1954. It should be remarked that the coefficient of utilization of useful volume is not employed abroad as a blast-furnace performance index.

For the purpose of comparison, the utilization of blast-furnace volume is set out below for the calender-year operation:

	1945	1955
U.S.S.R.	1.26	0.90 *
U.S.A.	1.55	1.19

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* Correcting for conversion pig-iron, the corresponding figures for the U.S.S.R. are 1.15 and 0.80.

It should be observed that compared with the U.S.S.R. furnaces, the U.S.A. furnaces have a somewhat narrower bosh with a large bosh and shaft angle.

The post-war period is marked by a wider application of carbon refractories for lining blast furnaces in the U.S.A. There are at present 102 blast furnaces with carbon lining in the U.S.A.

Large rectangular carbon blocks and small carbon bricks are employed in American blast furnaces. In Germany shorter blocks have come into use. Great Britain uses ribbed blocks and small carbon bricks.

Carbon brick-work in the center of the well without cooling does not provide sufficient durability. At present, therefore, attention is being turned to cooling the well and employing high thermal-conductivity carbon refractories in order to assist heat transfer from the well. There is a tendency to place carbon refractories around the periphery of the well in order to increase the depth of the cooling action of the coolers. Small size carbon bricks are the most suitable for this purpose.

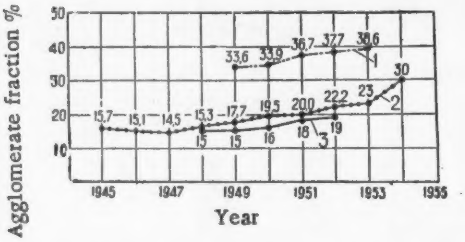


Fig. 4. Agglomerate fraction in the iron-ore part of the charge:
1) Federal German Republic; 2) Great Britain; 3) U.S.A.

Aluminosilicate brick is employed in the U.S.A. for lining the well, being of higher quality than in the U.S.S.R., having a higher alumina content, heat resistance and firing treatment and less porosity, with dimensions 460 x 150x 100 mm. Note might be made of blast-furnace well cooling from below, employed at plants in Czechoslovakia, Japan, Sweden and the German Democratic Republic, using air, water and oil as the cooling agent. In Czechoslovakia, Poland, the German Democratic Republic and the Federal German Republic, use is made of thin-walled shafts with spray-cooled jackets. A number of furnaces have been built without lintel girders and supporting columns. Operating experience on these furnaces, however, has proved unsuccessful in a number of cases.

Evaporative cooling of the tuyeres has been employed since 1955 in the Federal German Republic. The purpose was to reduce expenditure on the water supply system and to provide steam for district heating.

The post-war period has been marked by extended use of endless conveyers on blast furnaces. In 1949 transport of the charge to the blast furnace throat was carried out by a conveyer at the Trshinets plant in Czechoslovakia and in 1951 at the Cocquerille plant in Belgium.

In 1951 a distributor with accelerated rotation during discharge of the skip was employed at the Toledo plant in the U.S.A. At the Park Gate plant in Great Britain a cross-diameter and zig-zag diameter distribution cycle was introduced in 1952. Investigations have been carried out in the charge distributor cycle performance but without any definite results.

For screening coke fines, use is made abroad of a vibrating seive, which reduces grinding of the coke and the expenditure of replaceable elements.

As formerly, a feature of the U.S.A. is the use of large capacity hot-metal cars of the mixer type.

In the German Democratic Republic, for the purpose of reducing the length of the troughs, the iron is poured through one iron notch with a movable trough into travelling ladles, arranged in two rows.

In the U.S.A. cumulative charges are employed, for making up the iron notch; gas purification in a venturi pipe is commonly employed with subsequent cooling in the scrubber.

There is an increase abroad in dry filters using an asbestos and glass fabric. This method of gas cleaning is of great interest in connection with the utilization of the mechanical and thermal energy of the blast-furnace gas.

Of the new measurement and control devices note should be taken of instruments for hot-blast measurement, taking account of the blast losses in the air heaters (U.S.A.), and meters for measuring the dust content in the gas employing a photo-electric cell based on light absorption (Great Britain).

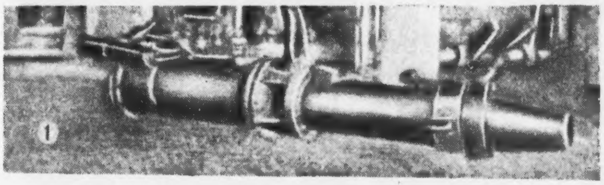
Automatic control of the blast based on pressure drop has been employed since 1951 in the U.S.A.

Radioactive indicators for checking the condition of the blast-furnace well should also be noted.

FROM THE TECHNICAL HISTORY

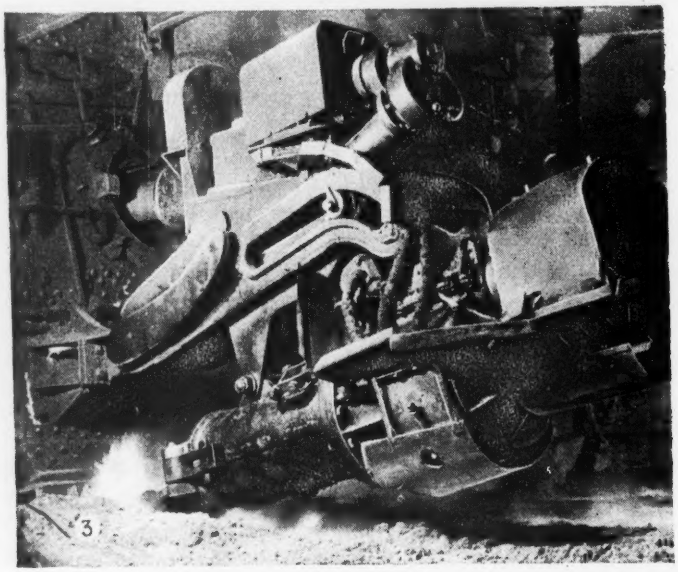
60 YEARS BACK

For a long time the iron notch on the blast furnace was sealed by hand after tapping the metal. The channel was rammed with clay and pressed home with special tools. In addition to the expenditure of heavy physical labor this operation required a long stoppage of the furnace.



The first device for closing the iron notch, called the clay gun, was put forward approximately sixty years ago—comparatively recently considering, that the history of development of the blast furnace goes back several ages.

This type of gun is illustrated in Fig.1. The back end of the gun carries a piston moving inside a cylinder under the action of steam. The piston was mounted on a bar which travelled along the front end of the gun, taking the form of a clay cylinder and ending with another cylinder. There was a hole (trap) in the clay cylinder through which the gun was charged with clay. The clay cylinder ended in a conical tip. The tip of the gun was inserted in the iron notch when it was necessary to seal it and clay was applied at the front wall of the hearth, constantly washed with slag which had flowed out with the iron. This tool, however, was inadequate. The single-cylinder gun did not obviate the necessity for almost complete withdrawal of the blast after each tapping. The gun was brought up to the iron notch manually. In addition, on applying the clay to the iron notch the blast-furnace men had to be near the furnace and as the piston was drawn away from the hearth, iron and slag frequently shot out.



The single-cylinder gun was replaced by an automatic two cylinder gun operated from one side on the Brosius system. With this gun it was possible to close the iron notch with full driving of the furnace without reducing the blast (note that at this time the blast pressure was considerably lower than at present).

The Brosius gun has now been superseded by the more advanced, reliable electric gun which is more satisfactory in operation.

The first steam gun in Russia was installed in 1911 at No. 3 blast furnace with a volume 400 m^3 at Yuzovka (former Stalin steel plant).

This gun is illustrated in Fig. 2, photographed immediately after installation. A group of blast-furnace workers is shown alongside the gun, at the head being Mikhail Konstantinovich Kurako (third from the right), the talented Russian blast-furnace inventor, who made considerable contributions to the development of Russian Metallurgy.

In 1956 the most advanced and powerful electric guns were installed in the blast furnace plant of the Magnitogorsk steel works, developing a pressure at the piston up to 180 t (Fig. 3). With the installation of this type of electric gun it is possible to seal the iron notch after tapping without reducing the blast and without affecting the productivity of the blast-furnace.

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February, 1957

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RUSSIAN AND GERMAN CRYSTALLOGRAPHY RESEARCH

IN COMPLETE ENGLISH TRANSLATION

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